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No. 67 JULY 1912

Bulletin of the American Institute of Mining Engineers.

CONTENTS:

SECTION I.—INSTITUTE ANNOUNCEMENTS.

PAGE

List of Officers and Committees, ... ii

Institute Announcements, ... iv

Library, ... xiii

Membership, ... xix

SECTION II.—PAPERS AND DISCUSSIONS.

Electrical Fume-Precipitation. By F. G. Cottrell, ... 667

The Solubility in Nitric Acid of Gold Contained in Certain Copper-Alloys (Copper-Bullions). By Edward Keller, ... 681

The Bearing of the Theories of the Origin of Magnetic Iron-Ores on Their Possible Extent. By Frank L. Nason, ... 695

Geology of Harrison Gulch, in Shasta County, California. By H. E. Kramm, ... 709

Abrasion and Dust-Losses in Ore-Drying. By Carl F. Dietz and Dyke V. Keedy, ... 717

A New Electric Miners' Lamp. By D. B. Rushmore, ... 739

An Early Discovery of Fullers' Earth in Arkansas. By J. C. Branner, 747

Contribution to the Study of Pre-Cambrian Rocks of the Harney Peak District of South Dakota. By Gordon S. Duncan, . . . 751

Discussion of Mr. Payne's paper, Progress in Roll-Crushing. By C. Q. Payne, ... 763

Discussion of Mr. Emrich's paper, The Refining of Blister-Copper. By Anton Eilers, ... 767

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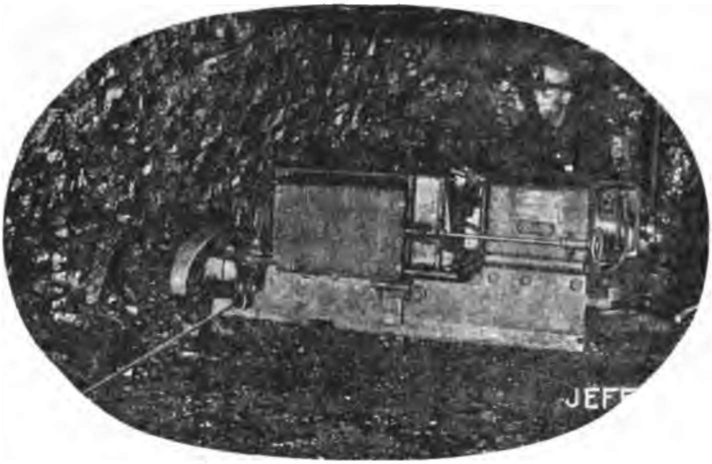
TECHNICAL JOURNALS AND OTHERS DESIRING TO REPUBLISH ARTICLES CONTAINED IN THIS *BULLETIN* SHOULD APPLY FOR PERMISSION TO THE SECRETARY, AT
29 WEST 39TH STREET, NEW YORK, N. Y.



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No. 67 JULY 1912

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TABLE OF CONTENTS.

SECTION I. INSTITUTE ANNOUNCEMENTS.

PAGE
List of Officers and Committees for the Year Ending February, 1913, . ii
Institute Announcements, ... iv
Library, ... xiii
Membership, ... xix

SECTION II. TECHNICAL PAPERS.

[The American Institute of Mining Engineers does not assume responsibility for any statement of fact or opinion advanced in its papers or discussions.]

No. 1. Electrical Fume-Precipitation. By F. G. COTTRELL, ... 667
No. 2. The Solubility in Nitric Acid of Gold Contained in Certain Copper-Alloys (Copper-Bullions). By EDWARD KELLER, ... 681
No. 3. The Bearing of the Theories of the Origin of Magnetic Iron-Ores on Their Possible Extent. By FRANK L. NASON, ... 695
No. 4. Geology of Harrison Gulch, in Shasta County, California. By H. E. KRAMM, ... 709
No. 5. Abrasion and Dust-Losses in Ore-Drying. By CARL F. DIETZ and DYKE V. KEEDY, ... 717
No. 6. A New Electric Miners' Lamp. By D. B. RUSHMORE, . . . 739
No. 7. An Early Discovery of Fullers' Earth in Arkansas. By J. C. BRANNER, ... 747
No. 8. Contribution to the Study of Pre-Cambrian Rocks of the Harney Peak District of South Dakota. By GORDON S. DUNCAN, . 751
No. 9. Discussion of Mr. Payne's paper, Progress in Roll-Crushing. By C. Q. PAYNE, ... 763
No. 10. Discussion of Mr. Emrich's paper, The Refining of Blister-Copper. By ANTON EILERS, ... - . 767

All communications concerning the contents of this Bulletin should be addressed to JOSEPH STRUTHERS, Ph.D., Secretary and Editor, 29 W. 39th St., New York, N. Y.

OFFICERS.

For the year ending February, 1913

COUNCIL.[*]

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(Term expires February, 1913.)

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R. W. RAYMOND	New York, N. Y.

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FRANK LYMAN, Treasurer; GEORGE C. STONE, Secretary;
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(Term expires February, 1914.)

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(Term expires February, 1915.)

Consulting Attorneys, Blair & Rudd, New York, N. Y.

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[*] Secretary's Note.—The Council is the professional body, having charge of the election of members, the holding of meetings (except business meetings), and the publication of papers, proceedings, etc. The Board of Directors is the body legally responsible for the business management of the Corporation.

COMMITTEES.

For the year ending February, 1913.

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Membership:—Joseph Struthers, Karl Eilers, Sidney J. Jennings, Benjamin B. Thayer, John H. Janeway, Jr.

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Land Fund:—James Douglas, *Chairman*; Theodore Dwight, *Treasurer;* T. F. Cole, Anton Eilers, James Gayley, John Hays Hammond, Charles Kirchhoff, Albert R. Ledoux, Frank Lyman.

INSTITUTE REPRESENTATIVES.

United Engineering Society Trustees:—Joseph Struthers (1913), Theodore Dwight (1914), James F. Kemp (1915).

Joint Conference Committee of the Four National Engineering Societies:—James F. Kemp, Joseph Struthers.

John Fritz Medal Board of Award:—E. Gybbon Spilsbury (1913), R. V. Norris (1914), Charles Kirchhoff (1915), J. F. Kemp (1916).

American Association for the Advancement of Science:—H. O. Hofman, John D. Irving.

Eighth International Congress of Applied Chemistry, New York, September, 1912:—William L. Saunders, George C. Stone.

Sixth Congress of the International Association for Testing Materials, September, 1912:—Robert Forsyth.

Committee No. 24, *International Association for Testing Materials*:—Henry D. Hibbard.

International Engineering Congress, San Francisco, 1915:—Samuel B. Christy, William C. Ralston, Edwin T. Blake.

Kelvin Memorial Committee:—James F. Kemp, Samuel B. Christy, James Douglas, Joseph Struthers.

Conference to Discuss Patent Laws:—A. F. Lucas, ————, ————.

INSTITUTE ANNOUNCEMENTS.

Special Notice.

The *Bulletin* is now entered at the Post Office at Second-Class Postage rate of one cent per pound, and in order to preserve this privilege it is necessary that the dues of members be paid within four months of January 1 of each year. If the dues are not paid within the period mentioned, a member's name must be removed from the regular subscription-list and the *Bulletin* mailed at the transient second-class postage rate of one cent for each four ounces or fraction thereof, prepaid by stamps affixed. It is therefore earnestly requested that dues be paid promptly—otherwise the Institute will be put to additional expense of postage and to added labor in removing and replacing names from the regular list, and in maintaining an additional separate mailing-list.

Adjourned Annual Business Meeting of the Institute.

At the Annual Business Meeting of the Institute, Feb. 20, 1912, it was voted to postpone the consideration of the proposed amendments to the Constitution until June 3, 1912, and the meeting adjourned to reconvene on that date, or on such subsequent date as should be fixed by the Special Committee of Five appointed on Feb. 20, and the Board of Directors of the Institute, acting jointly.

Due to various causes, including absence of two members of the Committee in Europe for two months, it has not been possible for the Special Committee of Five to complete its report in time for preliminary distribution to members of the Institute, and consideration by them prior to June 3, 1912, and it has been mutually agreed by the Special Committee and the Board of Directors to postpone the date at which the adjourned meeting shall be reconvened from June 3, 1912, to Oct. 7, 1912.

By order of the Board of Directors,

JOSEPH STRUTHERS,
Assistant Secretary.

Report of the Committee of Five.

The report of the Committee of Five, authorized at the Annual Business Meeting of the Institute, Feb. 20, 1912, to study the affairs of the Institute (see *Bulletin* No. 63, March, 1912, p. xviii.), has been received by the Board of Directors, and by them has been sent to the entire membership of the Institute. accompanied with a special letter of transmittal.

Proceedings of the Board of Directors.

The following acts of the Board of Directors at the meeting held June 28, 1912, are here published for the information of the members:

Additional appointments to the Iron and Steel Division Committee: Albert Sauveur and A. A. Stevenson.

Appointment of a special Committee to confer with the Committee of Five for mutual discussion of:

Proposed Amendments to the Constitution: Joseph W. Richards, James Douglas, and Hennen Jennings.

Appointment of special committees to report to the Board of Directors on:

Change of the By-Laws: Joseph Struthers, C. Kirchhoff, A. R. Ledoux, K. Eilers, and E. L. Young.

Life Members: Charles F. Rand, James Gayley, and George C. Stone.

Delinquent Members: Charles F. Rand, James Gayley, and George C. Stone.

Proceedings of the Council.

The following acts of the Council at the meeting held June 28, 1912, are here published for the information of the members:

The President announced the appointment of a Nominating Committee consisting of Philip N. Moore, John Birkinbine, D. W. Brunton, Seeley W. Mudd, Charles P. Perin, and Arthur L. Walker, to present names for officers of the Institute to be voted upon at the Annual Business Meeting, Feb. 18, 1913.

The President also announced the appointment of a Special Committee to report to the Council on:

Method of Printing the Publications of the Institute: Joseph Struthers, Charles Kirchhoff, and W. R. Ingalls.

Iron and Steel Division.

The Committee of the Iron and Steel Division is prepared to furnish to all members requesting it a list of those members of the Institute who are interested in iron or steel.

The work of this Committee has progressed rapidly and already 29 papers on iron and steel are being prepared for publication by the Institute, and we are confident that there will be a large attendance of members prominent in iron and steel affairs at the next meeting of the Institute.

The following is a list of papers promised:

The Heat-Treatment of Steel, papers by Henry M. Howe, R. R. Abbott, and Mark Ammon. (The two latter through the kind efforts of Prof. Charles H. Fulton.)

The Carbon-Iron Diagram, by Henry M. Howe.

A Novelty in Open-Hearth Furnace-Practice, by N. S. Maccullum.

Blowing-In of Blast-Furnaces, by Ralph H. Sweetser.

On Blast-Furnace Air-Pressures, by J. E. Johnson.

Charcoal Pig-Iron, by J. E. Johnson.

On Mayari Steel (author announced later).

Iron-Ore, by Dwight H. Woodbridge (further title later).

Method of Surveying and Sampling Diamond-Drill Holes, by E. E. White.

The Manufacture of Coke, papers by F. E. Lucas, W. H. Blauvelt, C. W. Andrews, and J. De Forrest.

The Influence of Titanium on the Strength of Iron Castings, by Bradley Stoughton.

The Iron-Mining Interests of Lake Superior, by Richard A. Parker.

Cuyuna Iron-Ore Range, by Walter A. Barrows, Jr., and Carl Zapffe.

The History of a Crucible Steel-Furnace, which has made an extraordinary record, by E. T. Clarage.

The Use of Thermit in Iron and Steel Metallurgy, by E. A. Beck.

The Concentration of Iron-Ores, papers by N. V. Hansell and R. H. Richards.

The Case-Hardening of Special Steels, by Albert Sauveur and G. A. Reinhardt.

The Heat-Treatment of Steel Castings, by C. D. Young.

The Present Status of Dry Blast, papers by James Gayley and Joseph W. Richards.

The Use of Schumacher Briquettes in the Johnstown Blast-Furnaces, by Felix A. Vogel.

Rational Cupola-Practice, by Richard Moldenke.

The Concentration- and Briquetting-Plant of the Moose Mountain, Limited, by Charles E. Herrmann.

An advance copy of such of these papers as are in print can be had by application to the Secretary of the Institute.

A generous response was made to the circular letter asking certain members to secure applications from their friends for admission to the Institute, and already a large number of applications have been received.

It is believed that what has already been accomplished bids fair to warrant the hope expressed in a former circular, to the effect that the usefulness and importance of the Institute in technical iron and steel matters shall become in the future even greater than in the past.

The co-operation of every member of the Institute is bespoken to secure as members all iron and steel men who would be desirable as such. The Committee further desires to maintain a correspondence with all interested members on the subjects referred to herein, and will greatly appreciate whatever encouragement it receives.

Yours respectfully,

CHARLES F. RAND, *Vice-Chairman.*
BRADLEY STOUGHTON, *Secretary.*

A New Volume on Ore-Deposits.

Ore-Deposits—a continuation of the "Posepny" Volume.

Comprising Papers Descriptive of Ore-Deposits and Discussions of their Origin, Edited, with an Introduction, by Dr. S. F. Emmons. The volume contains also a Biographical Notice of Dr. Emmons by his associate and friend, Dr. George F. Becker, and a comprehensive Bibliographical Index of the Science of Ore-Deposits, prepared by Prof. John D. Irving, of the Sheffield Scientific School of Yale University. Dr. Emmons had finished his editorial work and written his Introduction before his lamented death in 1910; and the Volume contains his last words upon the subject to which he had given the work of his life, and on which he was justly regarded as the foremost authority.

Further details concerning time of publication, price, etc., will follow.

Spokane Local Section.

The next meeting of the Spokane Local Section of the American Institute of Mining Engineers will be held at Republic, Wash., July 26, 27, and 28, 1912, invitation having been extended by the City of Republic through the Mayor of the City, the Commercial Club, the Mine Owners and our own members resident there.

L. K. ARMSTRONG,
Secretary.

Joint Meeting of the Institute and the International Congress of Applied Chemistry.

The joint meetings with Section III. A—Mining and Metallurgy—of the Eighth International Congress of Applied Chemistry, will be held at Columbia University, New York City, as follows:

Friday, Sept. 6, 10 a.m. to 12 m.; 1 to 3 p.m.
Topics: Metallurgy of Iron, Steel, Copper, Silver, Lead, etc.
Saturday, Sept. 7.
Topics: Electro-Metallurgy; in connection with Section "Electro-Chemistry" of the Congress.

Numerous lectures and entertainments on these and succeeding days are provided for members of the Congress only. Membership in the Congress costs $5, and entitles the members to all the proceedings of the Congress. The amount may be sent to William J. Matheson, Treasurer, 182 Front Street, New York City.

Certificate of Incorporation of the Institute.

By special request, the Certificate of Incorporation of the Institute is here republished for the information of the membership at large. This certificate was published in *Transactions*, Vol. XXXVIII., p. xvi., as a matter of official record, and it is also contained in the separate pamphlet of the Constitution and By-Laws of the Institute, which is sent to each new member with the notice of election.

CERTIFICATE OF INCORPORATION.

WE the undersigned, being all persons of full age and citizens of the United States, and a majority residents of the State of New York, desiring to form a corporation pursuant to the provisions of the Membership Corporations Law for the purpose of incorporating, as provided in Section 5 of Article I. of said law, the existing unincorporated association known as American Institute of Mining Engineers, do hereby make, acknowledge and file this Certificate for that purpose, and do Certify as follows:

I. That the American Institute of Mining Engineers is an unincorporated association organized and existing with the object of promoting the arts and sciences connected with the economic production of the useful minerals and metals and the welfare of those employed in these industries by means of meetings for social intercourse and the reading and discussion of professional papers, and to circulate by means of publications among its members and associates the information thus obtained.

II. That the persons duly appointed or designated to manage the affairs of said association are designated by the rules thereof Members of its Council; that the undersigned are all members of said Council as the same was constituted on the 29th day of December, 1904.

III. That on said last-mentioned date a regularly called meeting of said association was held at its office in the Borough of Manhattan, City of New York; that thirty days before such meeting notice of the intention to incorporate said association was given by mail to each member thereof whose residence or post-office address is known; that at said meeting the following resolutions were offered, seconded and duly adopted by the unanimous vote of all its members then present, to wit:

"*Resolved*, That it is the sense of the members and associates of the American Institute of Mining Engineers in general meeting assembled that it is desirable and necessary for the well being of said association and its members and for the furtherance of the objects for which the same has been formed, that said association incorporate under the Membership Corporations Law of the State of New York;

And Further Resolved, That the Members of the Council of this Association, or a majority thereof, be and they hereby are authorized, in accordance with the provisions of Section 5 of Article I. of the Membership Corporations Law, to incorporate this association for the same purposes for which it has been organized and conducted, in the manner provided in Article II. of said law;

And Further Resolved, That the name of said corporation as hereby adopted by this meeting shall be American Institute of Mining Engineers.

And Further Resolved, That the said incorporators shall be named in the Certificate of Incorporation as directors of such corporation until its first annual meeting, and that such directors and their successors in office shall be and they hereby are authorized to enact and adopt a Constitution and By-Laws for the government of said corporation."

IV. That the name of the proposed corporation is American Institute of Mining Engineers.

V. That the purposes for which this corporation is to be formed are: To promote the arts and sciences connected with the economic production of the useful minerals and metals and the welfare of those employed in these industries by means of meetings for social intercourse, and the reading and discussion of professional papers, and to circulate by means of publications among its members the information thus obtained; and to establish and maintain a place of meeting for its members and a hall for the reading of papers and delivery of addresses, and a library of books relating to subjects cognate to the sciences and arts of mining and metallurgy.

VI. That the territory in which the operations of this corporation are to be principally conducted is the United States of America, and the principal office for the transaction of its affairs is to be located in the City, County and State of New York.

VII. That the duration of this corporation is to be perpetual.

VIII. That the number of directors of this corporation is to be nine. At its first annual meeting three directors shall be elected to serve for one year; three directors shall be elected to serve for two years, and three directors shall be elected to serve for three years. At each subsequent annual meeting so many directors only shall be elected as shall be necessary to fill vacancies then existing in the Board; and each director then chosen shall be elected to serve for three years, unless the vacancy which he shall be chosen to fill was caused otherwise than by expiration of the former incumbent's term of office, in which event he shall be elected to serve for the balance of his predecessor's unexpired term.

IX. That the time for holding the annual meeting of this corporation shall be the third Tuesday of February in each year.

X. That the names and post-office addresses of the subscribers hereto who are to be directors of this corporation until the first annual meeting are

Names.	Post-Office Addresses.
James Gayley, ...	71 Broadway, New York City;
Frank Lyman, ...	88 Wall Street, New York City;
James F. Kemp, ...	Columbia University, New York City;
Charles H. Snow, ...	New York University, New York City;
Frank Klepetko, ...	24 State Street, New York City;
Thomas A. Rickard, . . .	261 Broadway, New York City;
James Douglas, ...	99 John Street, New York City;
Albert R. Ledoux, ...	99 John Street, New York City;
Rossiter W. Raymond, . .	99 John Street, New York City.

The above certificate, dated December 30, 1904, was signed and acknowledged by the subscribers before William A. Lockwood, Notary Public (No. 74), New York County.

I, the undersigned Justice of the Supreme Court of the State of New York, do hereby approve of the above Certificate of Incorporation of the American Institute of Mining Engineers, and do authorize the same to be filed.

SAMUEL GREENBAUM,
Justice of the Supreme Court.

January 7, 1905.

Library Research-Work.

The attention of members of the Institute is again directed to the research-work done by the librarian and his assistants, which should attract special attention from those members who have no access to the literature of subjects in which they may be interested.

During the year 1911 there were 143 searches made for members and non-members of the Founder Societies, and copies of the references have been preserved for the use of others. This work has been largely based on requests sent in by mail, from Japan, South Africa, Mexico, Canada, and England, as well as from different parts of the United States. The Librarian is confident that if it were more widely known that the library is equipped to undertake researches, the demand would increase beyond the ability of the present force to handle it. The library receives more than 700 technical periodicals which are available through the indexes for this special purpose.

Back Volumes of the Transactions.

The Board of Directors has authorized the following offers of sets of back volumes of the *Transactions*, at considerably reduced prices, to Members, Libraries, and Scientific Societies:

		Per Set.
I.	Five volumes, bound in half-morocco, from No. 36 (1906) to No. 40 (1910), ...	$20
II.	Ten volumes, bound in half-morocco, from No. 31 (1902) to No. 40 (1910), including Mexican Volume, . . .	35
III.	Twenty volumes, bound in half-morocco, from No. 21 (1893) to No. 40 (1910), ...	50
IV.	Thirty volumes, bound in half-morocco, from No. 11 (1883) to No. 40 (1910), ...	60
V.	Thirty-nine volumes, bound in half-morocco, from No. 1 (1873) to No. 40 (1910), with the exception of No. 10 (1882), but including index for Volumes Nos. 1 to 35, and Nos. 36 to 40, ...	75
VI.	Nine volumes, bound in half-morocco, from No. 1 (1873) to No. 9 (1881), ...	25

Applications should be addressed to Joseph Struthers, Secretary, 29 West 39th Street, New York, N. Y.

The Emmons Research Fellowship of Economic Geology.

The Committee named below has been formed by friends of Samuel Franklin Emmons, late of the United States Geological Survey, to consider the best method of perpetuating his name. It has been decided that the memorial to him shall take the shape of a Research Fellowship, to be known as the Samuel Franklin Emmons Research Fellowship of Economic Geology, which is to be administered by Prof. James F. Kemp, of Columbia University, New York. Subscriptions are invited by his friends to this fund, which the Committee has fixed at $25,000.

Members of the Institute who desire to contribute to this fund will please communicate with the Treasurer, Benjamin B. Lawrence, 60 Wall Street, New York.

The Committee consists of the following:

GEORGE OTIS SMITH, Director, U. S. Geological Survey, Washington, D. C.
H. L. SMYTH, Harvard University, Cambridge, Mass.
JAMES DOUGLAS, 99 John Street, New York, N. Y.
J. A. HOLMES, Director, Bureau of Mines, Washington, D. C.
JAMES F. KEMP, Columbia University, New York, N. Y.
F. W. BRADLEY, San Francisco, Cal.
J. PARKE CHANNING, 42 Broadway, New York, N. Y.
SEELEY W. MUDD, 1001 Central Building, Los Angeles, Cal.
D. W. BRUNTON, Denver, Colo.
H. FOSTER BAIN, 420 Market Street, San Francisco, Cal.
T. A. RICKARD, London, England.
B. B. LAWRENCE, 60 Wall Street, New York, N. Y.

Affiliated Student Societies.

Any society of undergraduates at a technical school, comprising students in any branch of engineering, metallurgy, chemistry, geology, etc., may be recognized by the Council in its discretion as an Affiliated Student Society. A circular giving details of the plan of affiliation may be obtained on application to the office of the Secretary of the Institute.

The following societies have been placed by authority of the Council on the above list:

AFFILIATED STUDENT SOCIETIES.

The Mining Society of the Sheffield Scientific School, Yale University, New Haven, Conn. *President*, Karl C. Stadtmiller; *Secretary*, S. B. Gordy.

The University of Illinois Student Branch of the American Institute of Mining Engineers, Champaign, Ill. *President*, Leonard V. Newton; *Secretary*, L. W. Swett.

The Engineering Society of the University of Nevada, Reno, Nev. *President*, D. E. Bruce; *Secretary*, R. M. Seaton.

The University of Wisconsin Mining Club, Madison, Wis. *President*, Rudolph J. Stengl; *Secretary*, Mack C. Lake.

The Mining and Geological Society of Lehigh University, South Bethlehem, Pa. *President*, William E. Fairhurst; *Secretary*, Carl W. Mitman.

The School of Mines Society of the University of Minnesota, Minneapolis, Minn. *President*, Emory P. Baker.

The Mining Engineering Society of the Massachusetts Institute of Technology. *President*, L. B. Duke; *Secretary*, Lionel H. Lehmaier.

The Student Auxiliary Society of the American Institute of Mining Engineers of the University of Kansas, Lawrence, Kan. *President*, A. H. Mangelsdorf; *Secretary*, C. J. Hainbach.

The Associated Miners of the University of Idaho, Moscow, Idaho. *President*, James W. Gwinn; *Secretary*, J. Wallace Strohecker.

The State College of Washington Mining and Geological Society, Pullman, Wash. *President*, H. E. Doelle; *Secretary*, B. R. Kinney.

The Tejas Technical Society, School of Mines, University of Texas. *President*, G. C. Cartwright; *Secretary*, David S. Alley.

The Ohio State University Student Branch of the American Institute of Mining Engineers, Columbus, Ohio. *President*, Hugh B. Lee; *Secretary*, E. P. Elliott.

The Stanford Geology and Mining Society, Stanford University, Cal. *President*, B. E. Parsons; *Secretary*, E. D. Nolan.

The Senior Mining Society of Columbia University, New York, N. Y. *President*, Roger L. Strobel; *Secretary*, Clark G. Mitchell.

Mining Association of the University of California, Berkeley, Cal. *President*, Frank L. Wilson; *Secretary*, Stanley L. Arnot.

Tufts College Chemical Society, Tufts College, Mass. *President*, P. G. Savage; *Secretary*, W. S. Frost.

University of Washington Mining Society, Seattle, Wash. *President*, Horace H. Crary; *Secretary*, Clinton R. Lewis.

Student Branch of the American Institute of Mining Engineers, Iowa State College, Ames, Iowa. *President*, M. B. Hadley; *Secretary*, R. L. Hurst.

Missouri Mining Association of the Missouri School of Mines, Rolla, Mo. *President*, D. L. Forrester; *Secretary*, J. S. Irwin.

The Pick and Shovel Club of the Case School of Applied Science, Cleveland, Ohio. *President*, L. B. Riddle; *Secretary*, S. C. Stillwagon.

Colorado School of Mines Scientific Society, Golden, Colo. *President*, Alan Kissock; *Secretary*, George Wilfley.

Mining Engineering Society of the University of Arizona, Tucson, Ariz. *President*, James J. Flanigan; *Secretary*, H. O. Coles.

How to Use the "Transactions" of the Institute.

Buy a copy of the Complete Analytical and Alphabetical Index of Volumes I. to XXXV., inclusive; also the new Index of Volumes XXXVI. to XL.

Whether you do or do not own a full set of the *Transactions*, these Indexes will make all of the material contained in the forty volumes available at once without detailed research into each volume separately. Moreover, an easy search will show what particular papers you need to know more about, and perhaps to study. Thus, any person possessing these Indexes can ascertain at once what has been published in the *Transactions* on a given question, and can learn, by writing to the Secretary, what is its nature, whether it is still to be had in pamphlet form, where it can be consulted in a public library, at what cost it can be copied by hand, etc., etc.

In short, to those who own complete sets of the *Transactions*, these Indexes will be a great convenience; but to those who do not, they will be a professional necessity.

The Index Volumes I. to XXXV. is an octavo of 706 pages, containing more than 60,000 entries, duly classified with sub-headings, and including abundant cross-references. The limited edition is becoming exhausted. The new Index, Volumes XXXVI. to XL., supplementing the Index Volumes I. to XXXV., brings the classified references up to the date of Volume XL., June, 1910. Prices: Index Volumes I. to XXXV., bound in cloth, $5; bound in half-morocco, to match the *Transactions*, $6. Index Volumes XXXVI. to XL., bound in cloth, $1.50; bound in half-morocco, $2.50. The delivery charges will be paid by the Institute on receipt of the above price.

LIBRARY.

AMERICAN INSTITUTE OF ELECTRICAL ENGINEERS.
AMERICAN SOCIETY OF MECHANICAL ENGINEERS.
AMERICAN INSTITUTE OF MINING ENGINEERS.
UNITED ENGINEERING SOCIETY.

WILLIAM P. CUTTER, Librarian.

The libraries of the above-named Societies are open from 9 A.M. to 9 P.M. on all week-days, except holidays, from September 1 to June 30, and from 9 A.M. to 6 P.M. during July and August.

The Library contains about 42,000 volumes, including sets of technical periodicals and the publications of scientific and technical societies.

The members of the Institute, with few exceptions, are by the very nature of their profession forced to spend a large portion of their time in localities isolated from sources of information. To such members the Library can render valuable service through correspondence, and letters requesting information will receive special attention. The Library is prepared to furnish references and copies of articles on mining and metallurgical subjects; to determine, if possible, the existence of mining-maps, and to furnish general information as to the geology and mineral resources of all countries as far as these resources are known and published.

It is hoped that the members of the Institute will avail themselves freely of this special service. The Library will welcome inquiries on engineering subjects, and furnish information as far as such information is to be obtained.

All communications should be made as definite as possible so that the information received may be what is desired and not include collateral matter which may not be of interest. In this way the time spent in searching for such collateral matter will be saved, and as a result the information will be sent more promptly and in more usable shape.

The members of the Institute can be of service to the Library by forwarding copies of mining-reports, maps privately issued, and similar material, which will be classified, indexed, and made available to other members.

Suggestions for additions to the Library, either by purchase or personal solicitation as gifts, will be welcomed. It is hoped that members while in the city will use the Library freely, and assurance is given that most careful service will be rendered to them.

Library Accessions.

June 1 to June 30, 1912.

[Copies of the list of additions to the Libraries of the American Society of Mechanical Engineers and the American Institute of Electrical Engineers can be obtained on application to the Secretary of the American Institute of Mining Engineers.]

AMERICAN INSTITUTE OF MINING ENGINEERS. Transactions. Vol. XLII. New York, 1912.

ASOCIACION DE INGENIEROS Y ARQUITECTOS DE MEXICO. ANALES. Tomo XIX. Mexico, 1912. (Exchange.)

BELGIUM. MINISTÈRE DE L'INDUSTRIE ET DU TRAVAIL. Statistique Rétrospective des Industries Extractives et Métallurgiques en Belgique, 1901–1910. Bruxelles, 1912. (Gift of Annales des Mines de Belgique.)

BERGWERKS INSPEKTION IN OSTERREICH, 1907, 1908. Wien, 1911, 1912. (Exchange.)

BOSTON. PUBLIC LIBRARY. Annual Report of the Trustees, 60th. Boston, 1912. (Exchange.)

CANADA. COMMISSION OF CONSERVATION. Lands, Fisheries and Game—Minerals, 1911. Ottawa, 1911. (Gift of A. R. Ledoux.)

CANADA. MINES DEPARTMENT. Investigation of the Coals of Canada. Vols. I.-II. Ottawa, 1912. (Exchange.)

CANADIAN MINING INSTITUTE. Journal. Vol. XIV. Montreal, 1911. (Exchange.)

CARNEGIE LIBRARY OF PITTSBURGH. Annual Report, 16th. Pittsburgh, 1912. (Exchange.)

CARTA GENERAL DE LA REPUBLICA MEXICANA. 1910. (Gift of Mexico Ministerio de Fomento.)

CHARTS OF ORGANIZATION OF ALL DEPARTMENTS OF THE CITY OF CHICAGO, AS IN EFFECT FEBRUARY, 1912. (Gift of Chicago Civil Service Commission.)

CHICAGO. BOARD OF SUPERVISING ENGINEERS CHICAGO TRACTION. Annual Report, 3d. Chicago, 1911. (Gift of Bion J. Arnold.)

CONGRESO CIENTIFICO (1° PAN AMERICANO). Ciencias Naturales, Antropologicas y Etnologicas. Tomo I. Santiago de Chile. 1911. (Gift of Congreso Cientifico, 1° Pan Americano.)

CONTRIBUTO ALLA NOMENCLATURA UNIFORME DEI PRODOTTI SIDERURGICI. Proposte, per la terminologia italiana, al IV Congresso della Association internazionale per le prove dei materiali. Milano, 1912. (Gift of Associazione fra gli Industriali Metallurgici Italiani.)

CUPOLA FURNACE, A PRACTICAL TREATISE ON THE CONSTRUCTION AND MANAGEMENT OF FOUNDRY CUPOLAS. By Edward Kirk. Philadelphia, 1910. (Purchase.)

DESIGN OF MINE STRUCTURES. By M. S. Ketchum. New York, 1912. Price, $4 net. (Gift of McGraw-Hill Book Co.)

[NOTE.—Apart from problems of economic geology and of ore-dressing, mine-engineering is simply civil, mechanical and (now-a-days also) electrical engineering, practiced in connection with mining, and therefore under peculiar conditions, especially those of work underground. Yet each of these departments of engineering has become so wide and so complicated that special treatises, dealing with its relations to mining, are needed, and therefore warranted. This book, for instance, is the work of a civil engineer, and deals with problems of civil engineering. The author, Professor of Civil Engineering in the University of Colorado, has already won professional recognition by his books on the design of walls, bins, and grain-elevators; the design of highway bridges and the calculation of stresses in bridge-trusses; the design of steel-frame buildings and the calculation of their stresses, etc.; and this work is another contribution in the same line, dealing particularly with such structures as the mining engineer is called upon to erect.

Under the head of mine-structures, Professor Ketchum includes head-works for mining-shafts; methods and apparatus of hoisting; and coal-tipples, breakers, and washers. But his treatment of these topics includes a thorough discussion of the calculation of strains in all such structures, and also in ordinary mine-buildings, including a consideration of timber structures, and—what is perhaps the most important and timely of all his contributions to current practice—a special consideration of reinforced-concrete structures. Not only theoretical formulas, but also suggested specifications, are given for each material.—R. W. R.]

DICTIONARY OF APPLIED CHEMISTRY. Vol. II. By Edward Thorpe. London, 1912. (Purchase.)

EARNING POWER OF RAILROADS, 1912. Compiled and edited by F. W. Mundy. New York, James H. Oliphant & Co., 1912. (Gift of Publishers.)

[NOTE.—This book contains chapters on Income Account; Operating Expenses; Maintenance Expenses; Traffic; Transportation and General Expenses; The Operating Ratio; Fixed Charges; Stock Outstanding in its Relation to Earning Power; Guarantees, and their Relation to Surplus Available for Dividends; and statistical tables for 154 railroads in the United States, Canada, Mexico, Cuba, and Panama. These figures are primarily useful to investors; but in these days, when everybody is attacking transportation-companies as the guilty factors in "social injustice" and the high cost of living, they may serve to furnish to some parties a legitimate support for their complaints, and to other parties a much-needed corrective of their reckless and ignorant statements.—R. W. R.]

EARTHQUAKES AT YAKUTAT BAY, ALASKA, IN SEPTEMBER, 1899. (Professional Paper No. 69, U. S. Geological Survey.) Washington, 1912. (Exchange.)

EINFÜHRUNG IN DIE MARKSCHEIDEKUNDE. By L. Mintrop. Berlin, 1912. (Gift of J. Springer.)

[NOTE.—This manual of 216 pp., with 191 illustrative diagrams and several colored plates, has been prepared, like so many other excellent text-books, to meet the requirements of actual experience in teaching. The author is an instructor in the well-known mining-school at Bochum, in Westphalia, and the purpose of his work is to furnish to prospective mine-foremen and officials a sufficient preparatory knowledge of the theory and practice of surveying to qualify them for the positions to which they may be called. While this purpose naturally leads to the treatment in special detail of the problems and methods of the Westphalian coal-field, it involves also a clear statement of the principles of measurement, calculation and graphic representation underlying all systems of mine-surveying and mine-mapping. The plan of the book might well be imitated in other manuals of its class; and, apart from this use, it conveys incidentally to all mine-surveyors an interesting and helpful view of the practice of their art as it is actually carried on in one of the most thoroughly equipped and most scientifically developed and exploited coal-mining districts of the world. The apparatus and operations represented in the leading collieries of Westphalia are sure to be, for their conditions, up-to-date, and worthy of consideration.—R. W. R.]

DIE ERZLAGERSTÄTTEN DER VEREINIGTEN STAATEN VON NORDAMERIKA. By C. L. Henning. Stuttgart, 1911. (Purchase.)

EXAMINATION OF PROSPECTS. A Mining Geology. By C. G. Gunther. New York, McGraw-Hill Book Co., 1912. (Purchase.)

GEMEINFASSLICHE DARSTELLUNG DES EISENHÜTTENWESENS. 8. Auflage. Dusseldorf, 1912. (Gift of Verein deutscher Eisenhüttenleute.)

GEOLOGY AND MINERAL RESOURCES OF THE PEORIA QUADRANGLE, ILLINOIS. (Bulletin No. 506, U. S. Geological Survey.) Washington, 1912. (Exchange.)

GRUNDZÜGE DES EISENHÜTTENWESENS. 1. Band. By Th. Geilenkirchen. Berlin, 1911. (Purchase.)

INSTITUTE OF METALS. Journal. Vol. VII. London, 1912. (Exchange.)

INTERNATIONAL CABLE REGISTER OF THE WORLD, 1912. New York, 1912. (Gift of International Cable Directory Co.)

JAHRESBERICHT ÜBER DIE LEISTUNGEN DER CHEMISCHEN TECHNOLOGIE FÜR DAS JAHR 1911. 1. Abteilung: Organischer Teil. Leipzig, 1912. (Purchase.)

JOHN CRERAR LIBRARY. Annual Report, 17th, 1911. Chicago, 1912. (Exchange.)

KOLLOIDCHEMIE EIN LEHRBUCH. By Richard Zsigmondy. Leipzig, 1912. (Exchange.)

KURZE ANALYSEN VORSCHRIFTEN. By Ernest Pancke. Halle a/S., 1912. (Purchase.)

LEHRBUCH DER CHEMISCHEN TECHNOLOGIE UND METALLURGIE. By Bernhard Neumann. Leipzig, 1912. (Purchase.)

MAP OF NORTH CAROLINA. 1911. (Exchange.)

METAL STATISTICS, 1912. New York, 1912. (Purchase.)

MINERALOGICAL NOTES, SERIES 2. (Bulletin No. 509, U. S. Geological Survey.) Washington, 1812. (Exchange.)

MINING LAWS OF THE STATE OF MONTANA. Compiled from the Revised Codes of 1907 and Session Laws of 1909–1911. N. p., 1911. (Exchange.)

MYSORE GEOLOGICAL DEPARTMENT. Report of the Chief Inspector of Mines, 1909–1910. Madras, 1912. (Exchange.)

DIE NASSBAGGER UND DIE BAGGEREIHILFSGERÄTE. By M. Paulmann und R. Blaum. Berlin, 1912. (Purchase.)

NEW JERSEY. GEOLOGICAL SURVEY. Annual Administrative Report of the State Geologist, 1911. (Bulletin No. 6.) Trenton, 1912. (Exchange.)

NEW SOUTH WALES. DEPARTMENT OF MINES. Annual Report, 1911. Sydney, 1912. (Exchange.)

ONTARIO. MINING ACT OF ONTARIO (1908) WITH AMENDMENTS TO 1912. Toronto, 1912. (Gift of Toronto Department of Forests and Mines.)

OPEN HEARTH STEEL CASTINGS. By W. M. Carr. Cleveland, 1907. (Purchase.)

PHILIPPINE ISLANDS. BUREAU OF SCIENCE. Annual Report, 10th. Manila, 1912. (Exchange.)

POLYTECHNIC ENGINEER. 1906, 1906–1907, 1912. Brooklyn, 1906–1907, 1912. (Gift of Polytechnic Institute of Brooklyn.)

[NOTE.—This interesting publication, issued annually by the undergraduates of the Polytechnic Institute of Brooklyn, contains many papers creditable to their authors and interesting to engineers. The recent liberal endowment of the Brooklyn Polytechnic, through the generous subscriptions of Dr. W. H. Nichols, President of its Board of Trustees, and his associates in that Board, together with other public-spirited citizens, has assured the success of an experiment which has been carried on for years, at great expense, and in the face of many discouragements, for the purpose of determining whether a technical school of the first class could be maintained in the Borough of Brooklyn. Dr. Nichols and President Atkinson are to be particularly congratulated on the way in which this victory has been achieved, not merely by soliciting financial support upon the strength of appeals to local pride and promises of great things to be done in the future, but by steadily continuing enlarging and improving the operations of the Institute, and proving practically, year after year, its acknowledged and growing efficiency. Its large faculty of instruction comprises many illustrious names; its courses are admirably arranged; and its students have the advantage of the personal attention and friendly aid of professors, to a degree not practicable in larger institutions. It has thus demonstrated beforehand that it well deserves and can effectively employ the new powers now bestowed upon it by those who have watched it closely and know it thoroughly, as an institution which not only teaches engineering, but makes engineers.—R. W. R.]

PROBENAHME UND ANALYSE VON EISEN UND STAHL. By O. Bauer and E. Diess. Berlin, 1912. (Purchase.)

QUEENSLAND. GEOLOGICAL SURVEY. Publication No. 231, 235. Brisbane, 1911. (Exchange.)

RHODESIA CHAMBER OF MINES. Annual Report, 17th, 1911. Bulawayo, 1912. (Exchange.)

TABLES ANNUELLES DE CONSTANTES ET DONNÉES NUMÉRIQUES DE CHIMIE, DE PHYSIQUE ET DE TECHNOLOGIE. Vol. I. Paris, 1912. (Purchase.)

TASCHENBUCH FÜR BERGMÄNNER. By Hans Höfer. 3d ed. 2 vols. L. Nüssler, Leoben, 1911. (Gift of Publisher.)

[NOTE.—The first edition of this famous little manual appeared 15 years ago. Since that time, the raw materials, methods and apparatus of mining have greatly changed, so that this third edition of Höfer's manual has had to be rewritten and enlarged, with the aid of many experts, and now appears in two volumes, covering together 1152 pages. Vol. I. contains chapters on mineralogy, geology and ore-deposits; prospecting, boring and stoping; mechanical mining; mine-workings; methods of exploitation; timbering and masonry; haulage and hoisting; ventilation; and the valuation of mining enterprises. Vol. II. treats of machinery for various purposes connected with mining; the mechanical preparation of ores and coal; coking; the briquetting of fuel and of ore; mine-surveying; and electrical engineering in connection with mines. In the *Engineering and Mining Journal* for Sept. 16, 1911, I expressed my judgment of both the limitations and the positive merits of this book, as measuring its value to American students and mining engineers. Referring to that review those who may desire a more critical estimate of it, I will here only repeat my recognition of its unquestionable professional value, and my congratulations of the veteran author upon this new evidence of his fruitful activity.—R. W. R.]

TASMANIA. GEOLOGICAL SURVEY. Mount Balfour Mining Field. Bulletin No. 10. Hobart, 1911. (Exchange.)

U. S. COAST AND GEODETIC SURVEY. Results of Observations made at the Coast and Geodetic Survey Magnetic Observatory near Honolulu, Hawaii, 1909 and 1910. Washington, 1912. (Exchange.)

U. S. GEOLOGICAL SURVEY. Bulletin Nos. 506, 509. Washington, 1912. (Exchange.)

UNIVERSITY OF PENNSYLVANIA. Engineering Alumni Society. Annual Report, By-Laws and Lists of Members. Philadelphia, 1912. (Exchange.)

VEREIN DEUTSCHER CHEMIKER e. V. Mitglieder Verzeichnis, 1912. Leipzig, 1912. (Gift of Verein deutscher Chemiker.)

WASHOE SMELTER. ANACONDA COPPER MINING CO., ANACONDA, MONT. N. p., n. d. (Gift.)

DIE WICHTIGSTEN LAGERSTÄTTEN DER "Nicht-Erze." Part I. By O. Stutzer. Berlin, 1911. (Purchase.)

DIE WIRTSCHAFTLICHE LAGE VON KANADA. By Hans Hammann. Berlin, 1912. (Purchase.)

DIE ZENTRIFUGALPUMPEN. Ed. 2. By Fritz Neumann. Berlin, 1912. (Purchase.)

GIFT OF ENGINEERING AND MINING JOURNAL.

AMALGAMATED COPPER CO. Report, 1912. New York, 1912.

ANACONDA COPPER MINING CO. Report, 1911.

CANADA. MINES DEPARTMENT. Investigation of the Coals of Canada. Vol. II. Ottawa, 1912.

DIESEL, RUDOLF. Present Status of the Diesel Engine in Europe and a Few Reminiscences of the Pioneer Work in America. St. Louis, 1912.

MINING JOURNAL (RUSSIAN). January, February, 1912. St. Petersburg, 1912.

MISSOURI. BUREAU OF MINES, MINING AND MINE INSPECTION. Annual Report, 23d, 1909. Jefferson City, 1909.

NOUMÉA. COMPTE-RENDU DES TRAVAUX DE LA CHAMBRE DE COMMERCE DE NOUMEA. Annee 1909, 1910. Noumea, 1910, 1911.

TRADE CATALOGUES.

SAMUEL CABOT, Boston, Mass. "Cabot" Conservo wood preservative. 14 pages.

CHICAGO PNEUMATIC TOOL CO., Chicago, Ill. Lubrication of Rockford motor cars. 11 pages.

GENERAL ELECTRIC CO., Schenectady, N. Y. *Bulletin No.* 4954. Horn type lighting-arresters for series lighting-circuits. 11 pages.

JEFFREY MANUFACTURING CO., Columbus, Ohio. Elevating-conveying, power-transmission machinery and coal-mine equipment. 576 pages.

STEPHENS-ADAMSON MANUFACTURING CO., Aurora, Ill. *The Labor Saver*, June, 1912. Published in the interests of efficiency and economy in production. 24 pages.

United Engineering Society Library.

HOW TO PLAN A LIBRARY BUILDING FOR LIBRARY WORK. By C. C. Soule. Boston, 1912. (Purchase.)

KENTUCKY. AGRICULTURAL EXPERIMENT STATION OF THE STATE UNIVERSITY. (Bulletin No. 154.) Blowing Stumps with Dynamite. Lexington, 1911. (Gift of Kentucky Agricultural Experiment Station.)

MOODY'S MANUAL OF RAILROADS AND CORPORATION SECURITIES. 13th edition, 1912. New York, 1912. (Purchase).

REPORT ON REVALUATION OF RAILROADS AND CANALS, NEW JERSEY, 1911. Trenton, 1912. (Gift of Charles Hansel.)

RULES OF PRACTICE IN THE UNITED STATES PATENT OFFICE. Revised July 17, 1907. Washington, 1912. (Gift.)

SVENSKT PORTRÄTTGALLERI ANDRA SERIEN. I. Svenska Teknologföreningen, 1911. Stockholm, 1911. (Gift of Svenska Teknologföreningen.)

"TITANIC" DISASTER. Report of the Committee on Commerce, United States Senate. Washington, 1912. (Gift of Senator W. A. Smith.)

TREATISE ON HYDRAULICS. By Mansfield Merriman. Ninth edition, revised and rewritten with the assistance of Thasseus Merriman. New York, J. Wiley & Sons, 1912. Price, $4 net. (Gift of Publishers.)

[NOTE.—This well-known work appears in a new dress, with much new matter. Nearly ten years have elapsed since the last complete revision, and now the book contains the results of the work of the last decade. Of the several editions of this work, 37,000 copies have been printed, the first edition being issued in 1889. The great increase in the development of the water-power plants in the country emphasizes the importance of a thorough revision of this standard work.—W. P. C.]

MEMBERSHIP.

NEW MEMBERS.

The following list comprises the names of those persons elected as members who accepted election during the month of June, 1912:

Members.

Barlow, Alfred E., Min. Geol......328 Roslyn Ave., Westmont, Que., Canada.
Bell, William L., Min. Engr., Supt. Smelter, B. C. Copper Co., Greenwood, B. C., Canada.
Braman, Harry S., Supt., Blast Furn., Youngstown Sheet & Tube Co., Youngstown, Ohio.
Clagett, Thomas H., Chief Engr., Pocahontas Coal & Coke Co., Bluefield, W. Va.
Crankshaw, Hugh M., Min. Engr........Lehigh Coal & Nav. Co., Lansford, Pa.
Crook, Welton J., Steel Wks. Chem., P. O. Box 985, Stanford University, Cal.
Flagg, Arthur L., Min. Engr.................212 Columbia Bldg., Spokane, Wash.
Gardner, E. D., Min. Engr............................Forest Service, Missoula, Mont.
Hurd, Walter E., Mine Operator...........................Haileybury, Ont., Canada.
Ingalsbe, Frank R., Min. Engr................. Forestry Service, Missoula, Mont.
Jones, Lewis M., Min. Engr...................U. S. Bureau of Mines, Pittsburg, Pa.
Manion, Edward, Leasing Mines............................221 S. Main St., Lead, S. D.
Morrison, Harold A., Mine Supt.......................................Candor, N. C.
Rightmire, Robert E., Fuel Engr., Consolidation Coal Co., Fairmont, W. Va.
Severy, Clarence L., Min. Engr., Poderosa Mining Co., Collahuasi, Chile, So. Am.
Siebenthal, Wade A., Supt. of Mines...................................Republic, Mich.
Sparks, James T., Min. Engr. and Mine Supt.............................Baldy, N. M.
Williams, J. Peter, Jr., Min. Engr...Tazewell, Va.
Yond, Herbert, Min. Engr...Guarda, Portugal.
Zalinski, Edward R., Min. Engr....607 Newhouse Bldg., Salt Lake City, Utah.

CANDIDATES FOR MEMBERSHIP.

The following persons have been proposed during the month of June, 1912, for election as members of the Institute. Their names are published for the information of members and associates, from whom the Committee on Membership earnestly invites confidential communications, favorable or unfavorable, concerning these candidates. A sufficient period (varying in the discretion of the Committee, according to the residence of the candidate) will be allowed for the reception of such communications, before any action upon these names by the Committee. After the lapse of this period, the Committee will recommend action by the Council, which has the power of final election.

Members.

Robert Rowell Abbott, Cleveland, Ohio.
Proposed by Charles H. Fulton, A. W. Smith, S. T. Wellman.

Mark Anthony Ammon, Cleveland, Ohio.
Proposed by Charles H. Fulton, A. W. Smith, S. T. Wellman.

Charles Woodward Andrews, Duluth, Minn.
Proposed by Charles F. Rand, Bradley Stoughton, F. Schniewind.

Abraham George Borowsky, Atco, N. J.
Proposed by Joseph W. Richards, Howard Ecksfeldt, Henry S. Drinker.

Charles Lang Cantley, New Glasgow, N. S., Canada.
Proposed by Thomas Cantley, Bradley Stoughton, John B. Porter, A. Stansfield, R. E. Chambers.

Alexander Fyfe, Cobalt, Ont., Canada.
Proposed by William Frank Grace. A. Jarman, Edwin G. Banks.

Frederick B. Goetter, San Dimas, Dur., Mexico.
Proposed by Edwin Lett Oliver, F. B. Caldwell, Stuart L. Rawlings.

Edward F. Goltra, St. Louis, Mo.
Proposed by E. Gybbon Spilsbury, Bradley Stoughton, Charles F. Rand.

Charles Henry Goodsell, Spokane, Wash.
Proposed by Glenville A. Collins, J. V. Richards, L. K. Armstrong.

William E. Hartman, Chicago, Ill.
Proposed by Bradley Stoughton, Charles F. Rand, Louis D. Huntoon.

Edward Hopkins Emerson, Santiago de Cuba, Cuba.
Proposed by Benjamin B. Lawrence, Charles F. Rand, DeBerniere Whitaker, M. B. Yung, Jennings S. Cox, Jr.

John Henry Klepinger, Great Falls, Mont.
Proposed by C. W. Goodale, A. E. Wheeler, John Gillie.

Edward Amos Loring, London, England.
Proposed by T. A. Rickard, C. S. Herzig, W. J. Loring.

Frank Edwin Lucas, Sydney, N. S., Canada.
Proposed by Charles F. Rand, Bradley Stoughton, F. Schniewind.

William Crooke McKee, Indiana Harbor, Ind.
Proposed by R. H. Sweetser, Joseph E. Thropp, Jr., J. H. Frantz.

Henry B. Newhall, Jr., New York, N. Y.
Proposed by H. M. Howe, Henry D. Hibbard, Bradley Stoughton.

William C. Perry, Cumberland, Md.
Proposed by B. F. Bush, C. H. Smith, J. M. Fitzgerald.

Wallace E. Pratt, Manila, P. I.
Proposed by F. A. Dalburg, Henry G. Ferguson, George I. Adams.

Alfred Clare Reed, Felton, Cuba.
Proposed by Charles F. Rand, George W. Pfeiffer, Jennings S. Cox, Jr.

Hans Egon Seidl, Cleveland, Ohio.
Proposed by Francis J. Peck, W. J. Rattle, G. F. Knapp.

William Robert Shimer, South Bethlehem, Pa.
Proposed by W. L. Cumings, E. O'C. Acker, R. M. Bird.

Henry Steinman Snyder, Bethlehem, Pa.
Proposed by Charles F. Rand, Bradley Stoughton, Jennings S. Cox, Jr.

John Venn Stevens, Panulcillo, Coquimbo, Chile, S. A.
Proposed by H. M. McLaughlin, Thomas F. Higgins, Mark R. Lamb.

Charles Allen Stillman, Chicago, Ill.
Proposed by D. B. Meacham, Charles F. Rand, Bradley Stoughton.

E. L. Tate, Spokane, Wash.
Proposed by L. K. Armstrong, J. V. Richards, F. A. Ross.

Edwin Harold Tatton, Oploca de Bolivia, S. A.
Proposed by Robert Hawxhurst, Jr., R. Gilman Brown, Charles Dalzell Wilkinson, J. H. Ivey.

William Joel Turner, Rancagua, Chile, S. A.
Proposed by R. T. White, Pope Yeatman, Bernard T. Colley.

Frank Earle Thompson, Johnstown, Pa.
Proposed by Hartley C. Wolle, Joseph Morgan, S. M. Marshall.

F. W. Wood, Sparrow's Point, Md.
Proposed by John E. Hardman, G. Bartol, Henry D. Hibbard.

Charles Duncanson Young, Altoona, Pa.
Proposed by George Thomas, 3d, Horace A. Beale, Jr., F. N. Pease.

Associate.

William A. Wilson, Spokane, Wash.
Proposed by L. K. Armstrong, J. V. Richards, Glenville A. Collins.

CHANGES OF ADDRESS OF MEMBERS.

The following changes of address of members have been received at the Secretary's office during the month of June, 1912. This list, together with the lists published in *Bulletin* Nos. 63 to 66, March to June, 1912, and the foregoing list of new members, therefore, supplements the annual list of members corrected to Feb. 1, 1912, and brings it up to the date of July 1, 1912.

BATESON, C. E. W.........830 Park Ave., New York, N. Y.
BOTSFORD, ROBERT S.........36 Leinster Sq., Bayswater, London, W., England.
BRETT, ALFRED J., Crown Mines, Box 102, Johannesburg, Transvaal, So. Africa.
BROWN, AUSTIN H.........“Walnut Grove Ranch,” Willits, Mendocino Co., Cal.
CONLEY, OWEN J.........52 Broadway, New York, N. Y.
COX, THOMAS.........463 Ellita Ave., Oakland, Cal.
DORION, FERDINAND J., Cons. Min. and Met. Engr., 24 Rue Alain Chartier, Paris, France.
DORR, JOHN VAN N.........733 First Natl. Bank Bldg., Denver, Colo.
DOZIER, CHARLES T.........P. O. Box 400, Redding, Cal.
DRAKE, FRANK.........Room 712, 79 Milk Street, Boston, Mass.
DUDLEY, HARRY C.........Instructed to hold all mail.
ELLIOTT, JOHN S.........61 E. 86th St., New York, N. Y.
EMMONS, N. H., Cons. Min. Engr.........917 Union Oil Bldg., Los Angeles, Cal.
FOOTE, F. S., JR., Asso. Prof. Railroad Engrg......Univ. of Calif., Berkeley, Cal.
GEPPERT, RICHARD M.........Salisbury House, London, E. C., England.
GOLDBERG, JULIUS.........Kamennoostrov Ave. 9, St. Petersburg, Russia.
GUITERMAN, FRANKLIN, Am. Smelt. & Refin. Co., 165 Broadway, New York, N.Y.
HAMILTON, ALEXANDER.........Dayton, Nev.
HAMLYN, JOHN J.........Natomas Cons. of Calif., Oroville, Cal.
HARRIS, ARTHUR L.........Rio Escondido, Coah., Mexico.
HEIMER, PAUL H.........498 Norfolk St., Mattapan, Mass.
JAQUES, W. H.........Little Boars Head, N. H.
JONES, CHARLES H.........28 Bourne Rd., Crouch End, London, N., England.
JONES, OWEN.........Instructed to hold all mail.
KAWAI, YASUSHIRO, Sakito Coal Mines, Sakitomura, Nishisonokigun, Nagasaki, Japan.
KELLER, ARTHUR H....Santa Rosa de Copan, Honduras, C. A., via New Orleans.
KEPNER, ROSS B.........Velardena, Dur., Mexico.
KIDDIE, THOMAS.........884 Bute St., Vancouver, B. C., Canada.
KINGSTON, CHARLES B., Care Pearce, Kingston & Browne, Worcester House, Walbrook, London, E. C., England.
KITSON, HOWARD W., Supt., Korncob Mining & Development Co., Redington, Pima Co., Ariz.
KRAMM, HUGO E.........P. O. Box E, Phoenix, Ariz.
LESLIE, HUGH M.........10 St. Andrews Drive, Pollokshields, Glasgow, Scotland.
LINTON, ROBERT, Min. Engr., Genl. Mgr., Sierra Cons. Mines Co., Trust & Sav. Bldg., Los Angeles, Cal.
MACCARTHY, MARION S.........4045 E. 19th Ave., Denver, Colo.
MANSFIELD, MELVIN.........Chino Copper Co., Hurley, N. M.
MAYNARD, T. POOLE, Min. and Cons. Geol., 321 James Bldg., Chattanooga, Tenn.
MERRILL, MONROE E.........Santa Rita, N. M.
MILTON, MAXWELL C.........6 Schrader Bldg., Tucson, Ariz.
MORAN, ROBERT B.........311 California St., San Francisco, Cal.
NICHOLLS, JOHN C.........1043 Mills Bldg., San Francisco, Cal.
PAGLIUCHI, FRANK D.........1144 Title Insurance Bldg., Los Angeles, Cal.
PEARCE, FRANK E.........Paulson, B. C., Canada.
PENTLAND, WALTER J., Mgr. Amajac Mines Co., Hostotipaquilla, Jal., Mexico.
PHILLIPS, WALTER I.........Hurley, N. M.
POCOCK, CECIL W., Mgr.........Santa Clara Mines, Morococha, Peru, So. Amer.
PRENTIS, EDMUND A., JR.........408 W. 23d St., New York, N. Y.
PROUST, RENÉ.........Quala Lumpur, Federated Malay States.
ROBBINS, CHARLES P.........P. O. Box 1910, Spokane, Wash.
RODGERS, CHARLES E., Min. Engr. and Assayer, Motherlode Mill, Sheep Creek, B C., Canada.

Schneider, Albert F.......132 Hodge Ave., Buffalo, N. Y.
Smith, Lyon.......P. O. Box 85, Blair, Nev.
Spicer, Philip O.......P. O. Box 544, New Westminster, B. C., Canada.
Stoutz, Edmund de.......Singapore, Straits Settlements.
Stuckey, Leonard C., Tauquah Min. & Expl. Co., Ltd.,
Tarquah, Secondee, W. Africa.
Talcott, Morris G., Care General Assets, Ltd., Dominion Bond Bldg.,
Toronto, Ont., Canada.
Thomas, David R.......Pridilecta Mining Co., 55 Liberty St., New York, N. Y.
Timmons, Colin.......4819 Budlong Ave., Los Angeles, Cal.
Truschkoff, Nicolas.......Kuokkala, Finlande Railway, Russia.
Venables, Harry L.......Oruro, Bolivia, So. America.
Villadsen, Anders B.......Care Pingree National Bank, Ogden, Utah.
Wallis, H. Boyd.......48 Holland Park, London, W., England.
Warner, Thor.......Hotel Golden, Reno, Nev.
Wickware, Francis G.......35 W. 32d St., New York, N. Y.
Williams, Ralph B.......The "Argyle," San Antonio, Texas.
Wishon, A. Emory.......San Joaquin Light & Power Corpn., Bakersfield, Cal.
Willson, Frederick E.......Santa Barbara, Cal.
Wolf, Albert G.......1574 Steele St., Denver, Colo.
Wright, Charles W.......Ingurtosu, Arbus, Sardinia, Italy.
Zeller, Howard P.......Care Republic Iron & Steel Co., Republic, Pa.

ADDRESSES OF MEMBERS AND ASSOCIATES WANTED.

Name.	Last Address of Record, from which Mail has been Returned.
Baxter, Francis K., Jr.,	423 Wells Fargo Bldg., San Francisco, Cal.
Chamberlain, John R.,	3a de Roma 39, Mexico City, Mexico.
Cook, Edward H.,	Minas Birimoa, S. A., Birimoa, via Canelas, Dur., Mexico.
Danforth, A. H.,	Cotopaxi, Colo.
Fitzgerald, Thomas F. M.,	211 Sharon Bldg., Salt Lake City, Utah.
Furness, James W.,	Coffee, Trinity Co., Cal.
Gmelin, Ernest,	Apartado 57, Aguascalientes, Mexico.
Green, George G.,	133a Vermont Ave., Washington, D. C.
Hamfeldt, George K.,	Care de Wendel & Co., Hayingen, Lothringen, Germany.
Higgins, Edwin,	Randsburg, Cal.
Hollis, R. W.,	Silverton, Colo.
Johnson, Dion L.,	325 Water St., Pittsburg, Pa.
Lampshire, John O.,	Vulture Mine, Wickenburg, Ariz.
Leavell, John H.,	Buffalo Mine, Cobalt, Ont., Canada.
Le Noir, Frank H.,	Box 16, Mt. Bullion, Cal.
McDougall, Wallace D.,	20 Bedford Place, Russell Sq., London, Eng.
Macfarlane, Rienzi W.,	Apartado 102, Parral, Chih., Mexico.
Moore, Roy W.,	P. O. Box 48, Velasco, Tex.
Munroe, Martin,	Bengal Coal Co., Murulidih, Mohada, B. N. Ry., Bengal, India.
Nelson, D. W. C.,	Baker City, Ore.
Nobs, Frederick W.,	Negociacion Minera Santa Maria de Guadalupe y Anexas, S. A., Minillas, Zac., Mexico.
Pearson, William R.,	628 W. 114th St., New York, N. Y.
Perks, Harry B.,	419 Board of Trade Bldg., Portland, Ore.
Peterson, Frank,	H. W. Hellman Bldg., Los Angeles, Cal.
Rathborne, Merwyn R. W.,	Amargosa, via Las Vegas, Nev.
Rhew, James W.,	Cia. Minera y Exploradora de Ventanas, S. A., Ventanas, Dur., Mex.
Sheldon, Waldo,	Urique, Chih., Mexico.
Short, Frank R.,	Carson City, Nev.
Thornton, Edward T.,	Apartado 30, Matehuala, S. L. P., Mexico.
Twynam, Henry,	O. K. Copper Mine, Cairns, No. Queensland, Australia.
Watson, Ralph W.,	Calloo, Utah, Clifton Mail box.
Webster, Erastus H.,	Hotel Cosmopolita, Guadalajara, Jal., Mexico.
White, W. P.	10 Bernard St., Spokane, Wash.

NECROLOGY.

The death of the following Honorary Member was reported to the Secretary's office during the month of June, 1912:

Date of Election.	Name.	Date of Deceese.
1899.	Osmond, Floris, ...	———, ——.

BIOGRAPHICAL NOTICES.

William Affleck was born July 26, 1880, at Liverpool, Eng. In 1903, he came to the United States, and was employed as civil and mining engineer by the James W. Ellsworth Co., at its coal-mines, Ellsworth, Pa. In 1906, he became an assistant to Mr. Ellsworth, in New York City, and, during the next two years, traveled extensively through Germany and other countries, gathering data of the coking and briquetting of coals and lignites. In 1908, he was sent by Mr. Ellsworth to the Cambridge Collieries, near Cambridge, O., and in 1909, he became General Superintendent of the Cambridge Collieries Co. In 1910, he was appointed General Superintendent of the holdings in the United States of the Sterling Coal Co., Ltd., Toronto, Can. This position he held until his death, which occurred at Cleveland, O., Sept. 11, 1911, after a long illness due to typhoid fever. Mr. Affleck became a member of the Institute in 1906. He was beloved and respected; and his death, at 31 years of age, is sincerely lamented as the premature end of a useful and promising career.

Ernest Robertson Buckley was born Sept. 3, 1872, at Milbury, Mass., educated in the district and high schools of Tomah, Wis., and graduated as B. S., in 1895, from the University of Wisconsin, which gave him, in 1898, the degree of Ph.D. For some time after his graduation, he was a member of the staff of the Wisconsin Geological and Natural History Survey; and in 1901, he became Director of the Missouri State Bureau of Geology and Mines and State Geologist of Missouri, with headquarters at Rolla. This office he retained until 1908, when he accepted the position of Economic Geologist of the Federal Lead Co. of Missouri. Such employment, by large railroad, mining, and land companies, of trained geologists to guide their operations, is a modern development, which has already abundantly proved its merit by its results. After two years of work in this capacity, Dr. Buckley devoted himself chiefly to consulting work and to travel in the Western States. In 1911, he opened an office as consulting engineer in Chicago.

Dr. Buckley was a life-member of the Institute, elected in 1902, and contributed to the *Transactions* (xl., 856) an interesting discussion of the Ozark lead- and zinc-deposits, their genesis, localization, and migration. His numerous published works comprise reports on the Building- and Ornamental-Stones of Wisconsin (1898); Clays and Clay-Industries in Wisconsin (1900); Highway Construction in Wisconsin (1902); and The Quarrying-Industry of Missouri (1904), besides other monographs and general discussions contained in the volumes of the Wisconsin Geological and Natural History Survey, and the Missouri Bureau of Geology and Mines. He was a member of the Geological Society of America; the American Mining Congress; the Wisconsin Academy of Sciences, Arts, and Letters; the Wisconsin Clay-Workers' Association; the Wisconsin Natural History Society; the American Society for Testing Materials; and an honorary member of the Black Hills Mining Men's Association. His death, Jan. 19, 1912, in his fortieth year, under an attack of malignant pneumonia, was equivalent, in its shock of surprise to his friends, to a violent death by accident. It cut short a career of industry, intelligence, high promise, and distinction already achieved.

R. W. R.

John Charles Dods was born in 1846, at Spalding, Lincolnshire, Eng., where his family had resided for several centuries. After a preliminary education and practice, of which we have not been able to obtain the record, he came to the United States, in 1870, as engineer for an English firm, with headquarters at Boston, Mass., and was engaged in erecting gas-furnaces in steel- and iron-works throughout the United States for a number of years. After employment in several cities of Pennsylvania, and in Chicago, he became (about 1890 or earlier) draftsman and engineer for the Laclede Fire-Brick Co., of St. Louis, Mo. He retained this position for fourteen years. In 1902, he was engaged to plan and erect large smelting-works for zinc, etc., at Cherryvale, Kan., and in 1904, he accepted the superintendency of these works, which he continued to manage until his death, which occurred Sept. 1, 1911, after a painful illness.

Mr. Dods had been a member of the Institute for 30 years, having been elected in 1881. He was a member of the National Gas Institute also, and was highly esteemed, both by his professional associates and by his fellow-citizens. His death evoked from the community in which he resided an impressive tribute of grief and praise.

John B. Hobson was born in Ireland about 1844. In 1848, his parents came to New York, and in 1859 they removed to California, where he studied metallurgy and mining engineering, and became connected with many of the large quartz-mining and deep-channel hydraulic operations of that State. As a field-contributor to the geological department of the State Mining Bureau, he furnished to its annual reports much interesting and valuable material. In the field of practice, he distinguished himself by the invention of a deflecting hydraulic nozzle, known and highly esteemed by miners as "Hobson's Improvement." In 1891, Mr. Hobson was one of the leading mine-operators who met at Auburn, Cal., and formulated plans for the California Miners' Association, fully organized in 1892, which has done so much for that industry. Hon. Jacob H. Neff, afterwards Lieutenant-Governor of California, was present at that preliminary meeting; and in 1899, Mr. Neff contributed to the magnificent souvenir volume, *California Mines and Minerals*, presented to members of the Institute party attending the California meeting of that year, an interesting account of the history of that famous association, of which he was at that time the President; W. C. Ralston, Vice-President; and Edward H. Benjamin (the editor of the volume referred to), Secretary. The Association had grown in seven years to a membership of 9,000. Mr. Hobson, who, as one of its founders, and a member of its Legislative Committee, did so much for it, and who had become, in 1892, a member of the Institute also, was not present at the California meeting, to receive, with his associates, our congratulations upon the result of their wise and efficient labors. Already in 1891 or 1892, he had been invited by Sir Wm. Van Horn and the Directors of the Canadian Pacific Railway Co. to visit British Columbia, and make a reconnoissance of the auriferous deep gravels of that Province. His investigations and reports led to the organization of at least two important hydraulic mining companies, the Consolidated Caribou and the Horsefly, and to his permanent removal, in connection with these and other enterprises, to British Columbia, where he resided for the rest of his life.

In his new home, Mr. Hobson at once assumed the leadership due to his ability and experience. He brought about the formation of a Provincial Miners' Association, like that of California; and at the convention called for this purpose, he was earnestly nominated as President of the Association, on the ground that he was practically its founder. But it was finally decided (probably on his own advice) that it would be wiser not to choose a President specially identified with a particular branch of mining, as Mr. Hobson was identified with placer and hydraulic gold-mining. So they elected a gentleman not actively engaged in any department of the industry. Whether this prudent decision had anything to do with the subsequent history of the Association, I will not undertake to say; but, according to the statement of the Vancouver *Daily Province*, of Jan. 20, 1912, the Provincial Mining Association, after a promising start, failed utterly in two or three years, to the great disappointment of Mr. Hobson, who, moreover, spent nearly $1,000 in paying the debts it had incurred.

As manager of the Horsefly and Caribou companies, Mr. Hobson directed a large amount of development, and produced a considerable quantity of gold. But his plans required the further expenditure of about $300,000 for an adequate and permanent water-supply. Particulars can be found in the reports of the Provincial Geologists, and his own reports to his employers. When, in 1906, the Caribou property was purchased by the Guggenheims, of New York, it was expected that this additional expenditure would be made at once. But in July, 1907, the new owners suspended operations, and work has never been resumed. It is reported that the financial stringency of 1907, which caused the abandonment by the Guggenheims of many enterprises not actually profitable at the time, was the reason of this sudden change. Indeed, it is scarcely conceivable that any weakness in the enterprise itself could have been the cause; for the courageous and experienced capitalists referred to had had the merits of the scheme thoroughly examined by their own experts before engaging in it; and no new evidence had been developed. Their decision was a shock to Mr. Hobson, from which he, perhaps, never fully recovered. He became seriously ill, and for some time his life was despaired of. After his recovery, he continued to reside at Victoria, B. C., where he died Jan. 9, 1912.

Mr. Hobson was one of the best authorities on the methods and appliances of hydraulic mining. Yet, apart from his early contributions to the reports of the California Mining Bureau, I find no record of anything from his pen except a paper on The Auriferous Gravels of British Columbia, published in the *Journal of the General Mining Association of Quebec*, vol. ii., p. 177. In 1907, Mr. John Hays Hammond, then President of the Institute (who is said to have acquired some of his first experience in mining under Mr. Hobson), united with me in urging him to prepare a paper for the Institute on that subject, concerning which our *Transactions* had contained little of comprehensive importance since the admirable treatise of A. J. Bowie, Jr., in 1877 (*Trans.*, vi., 27). But the request was unavailing, for reasons not known to us at the time, but easily to be inferred from the foregoing story, which shows that the time when our appeal came to him was the time of his greatest business anxiety and stress, and his consequent illness.

Besides Mr. Hobson's membership of the Institute, and of the Mining Associations of California and British Columbia, already mentioned, he was one of the founders of the Canadian Mining Institute, chartered, in 1898, by the Dominion Parliament. In 1898 and 1899, and again in 1901 and 1902, he was a Councilor, and in 1903 and 1904, a Vice-President, of that Institute. R. W. R.

Henry Lakin Lawrence was born May 28, 1859, at Wimbledon, near London, Eng. From 1876 to 1880, he studied at the Royal Mining Academy of Freiberg, Saxony. In 1881–83, he served as an "articled assistant" to a London civil engineer. In 1884–85, he was Assistant Manager of the El Bote Mining Co., Zacatecas, Mex. In 1885–87, he occupied a similar position with the Disraeli Mine Syndicate Co., at Charters Towers, Queensland. In 1888–89, he was engaged in practice as consulting engineer at Melbourne. In 1890, he became manager of the Chamelicon mines, in Honduras, Central America. In 1891–93, he was manager of the New Pierrefitte mines, Hautes Pyrénées, France. In 1894–95, he was manager of the Metropolitan mine, Johannesburg, South Africa. In 1895–96, he was a consulting engineer in Rhodesia. From 1897 to 1901, he was employed in the exploration of concessions bearing gold, diamonds, or manganese, in Brazil. In 1902, he conducted a surveying and prospecting expedition to Ashanti and the "Gold Coast" region of Africa. In 1903–05, he was manager of several gold-mining companies in the Transvaal. In 1906–07, he conducted an expedition of mining exploration to Northeast Siberia. In 1908, he was peacefully settled as a consulting engineer in his London office.

To the casual reader, the foregoing sketch of a "whirlwind" career, never stopping for more than a year or two at any one point, might seem like the story of a rolling stone, gathering no moss. But he who knows the ways of British investors will divine that this man, having earned the confidence of a circle of enterprising capitalists, was sent by them, in swift succession, to all parts of the world—here to reconnoiter in advance, there to organize and initiate, their mining adventures. The story, as I have told it above, leaves Mr. Lawrence in his London office. I am not surprised to learn that he was sent later to inspect mining properties in Norway. Unfortunately, his health, seriously impaired by the hardships of the Siberian exploration of 1906–07, was further imperiled by a serious illness, contracted in Norway, in Dec., 1910, from which he never fully recovered, although he managed, after this attack, to resume his work. He died of heart-failure, May 8, 1911, on his way home from the last of his many adventurous expeditions.

Mr. Lawrence had been a member of the Institute since 1893. When he died, at the age of 52, the profession to which he belonged lost not only an accomplished member, and the prospect of much excellent work which he was qualified to perform, but also a vast amount of useful knowledge and experience which he had accumulated, and which died with him. Might not an old man of 72 preach a little sermon on this text to the young ones of 52—or even less—or even more? R. W. R.

Charles A. Matcham was born Jan. 15, 1852, at Torquay, Devonshire, Eng., and educated in the schools of Hambledon and Brighton. Obtaining employment, at the age of 23, in a London engineering business, he attended also a night-class in engineering, winning, after public examinations, the first prize. In 1879, he entered the service of the American Bell Telephone Co., which had then recently obtained concessions for the establishment of telephone exchanges in many European countries. He built exchanges and organized operations in Antwerp, Brussels and Charleroi, Belgium, and also in Riga and St. Petersburg, Russia, where the Czar's winter-palace was wired under his direction—an operation which greatly promoted the introduction of the telephone throughout the Russian Empire.

In 1881, Mr. Matcham came to the United States, and continued his work as a builder of telephone exchanges at Chicago, Ill., Memphis, Tenn., and Allentown, Pa., where in 1884 he became, and remained for several years, manager of the Pennsylvania Telephone Co.

In 1890 Mr. Matcham associated himself with his brother-in-law, Thomas D. Whittaker, under the name of the Whittaker Cement Co., establishing a cement-works near Phillipsburg, N. J., of which he was General Superintendent until, and after, the plant was sold to the Alpha Portland Cement Co., in 1893. Resigning in 1897, he organized, with Col. H. C. Trexler, the Lehigh Portland Cement Co., the plant of which he managed for a number of years. More recently, he erected the mill of the Allentown Portland Cement Co., at Evansville, Berks county, Pa.; and he was President of the Fuller Engineering Co. until, in 1910, he was compelled by illness to retire from business activity. The number of successful enterprises which he organized and conducted proves not only his ability, but also the high degree of confidence reposed in him by investors. Some of the cement-machinery employed in these enterprises was invented by him.

Besides this Institute, of which he became a member in 1897, Mr. Matcham was a member of the American Society of Civil Engineers, the American Society for Testing Materials, the National Geographical Society, the Manufacturers' Club of Philadelphia, and other organizations in Allentown and Easton. He died, Sept. 16, 1911, greatly lamented by his fellow-citizens and business associates, as well as the closer circle of kinship and friendship. In view of his leadership in the great modern industry of the Lehigh Valley, the Allentown *Morning Call* gave to its editorial tribute a most appropriate title—"The Passing of a Master Builder."

R. W. R.

Herman C. Meister was born June 22, 1855, at St. Louis, Mo. In 1872, after preliminary education at a private school and at Smith Academy, he entered Washington University, from which he was graduated in 1876 with high honors, receiving the degree of E. M. During the following term, he was an Instructor at the University, and in 1877 he went first to direct prospecting-work in southwestern Missouri, and later to Leadville, Colo., where he became assayer for a silver-mining company. In 1878, he joined a prospecting-party in Mexico and Lower California; but the exertions and hardships of this trip broke down his health, and he returned to St. Louis in 1879 for needed rest and recovery. In 1880 he went to Joplin, Mo., where he was professionally employed until 1883, when he became connected with the zinc-smelting works at Collinsville, Ill. These works had been established in 1879, and were owned by private firms until 1888, when the business was incorporated as the Collinsville Zinc Co., of which Mr. Meister was for many years President. I quote from the *Engineering and Mining Journal* of Dec. 30, 1911, the following description of these works (which were operated with profit for many years) and of Mr. Meister's contributions to metallurgical progress:

> It was a plant of Belgian furnaces, which, at the time of its construction and for many years subsequently, were of almost universal use in the West; but the Collinsville plant was superior to most of the constructions of its time, evidencing the thought and direction of a well trained and scientific metallurgical engineer. This was the first zinc-smelting plant in the United States to be built entirely above ground (save, of course, the foundations), thus dispensing with the unsanitary and uneconomical subterranean furnace-cellars that used to characterize zinc-smelting works, not only in the United States but also in Europe. Mr. Meister, jointly with Mr. Edgar, of the Glendale works, St. Louis, was the first to introduce mechanical reverberatory furnaces for blende-roasting in the United States, and for many years the Brown horseshoe-furnace was successfully in use at Collinsville. These few examples indicate Mr. Meister's intelligence and enterprise in the zinc-smelting business. In 1901, or about that time, when the old Belgian coal-smelters had generally to be put out of operation, owing to inability to compete successfully with the smelters using natural gas, the Collinsville works also had to become idle. However, they were maintained in good repair, and from time to time were put in operation for short campaigns when market-conditions enabled a profit to be made. Thus, they were in operation in the early part of 1911.

The courage and skill with which Mr. Meister maintained for so many years a losing fight against hostile natural conditions reflect great credit upon him. In 1877, he became a member of the Institute; and in 1904, he presented, at the St. Louis supplementary session of the Lake Superior meeting of that year, a paper on The Zinc-Smelting Industry of the Middle West (*Trans.*, xxxv., 734), which exhibited his comprehensive grasp of the subject, and his acquaintance with existing conditions and practice. It is worthy of notice that nothing in this paper favors, or even indicates, the personal interest of the author in any particular process or enterprise. The whole of it might have been written by a disinterested outside observer—only an outsider would not be likely to know so much about the inside!

The renewal of work at Collinsville in 1911 (the fatal rivalry of natural gas in other districts having largely died out) was probably too much for Mr. Meister's strength. The favorable conditions for which he had so long waited had come at last; but one condition had become less favorable, namely, his own ability to carry the burden. At all events, on the morning of Nov. 27, 1911, having risen and breakfasted early, in order to meet a business engagement, he suddenly became unconscious, and died in the presence of his devoted wife, before the physician could be of use. A subsequent *post-mortem* examination revealed, as the cause of death, "arterial sclerosis," which is the inevitable sign and accompaniment of the failure due to advancing age; and age in a man, as in a blast-furnace, is measured, not by years of duration, but by the tonnage of product and the total of wear. Thus, at the age of 56, this strong man, having done a full life's work, fell in harness, while still straining at his work.

Mr. Meister's personal integrity, generosity, loyalty in love and friendship, and devotion to duty, were characteristics which should be recognized in this brief and imperfect notice. R. W. R.

Paul Ambrose Oliver was born July 18, 1831, in the English Channel, on board of an American vessel, of which his father was the commander. After a preliminary education in Germany, he returned to New York, and engaged in the shipping business, which he subsequently limited, in partnership with his brother, to the cotton-trade between New Orleans and New York. During this period, he resided at Fort Hamilton, N. Y.; and in 1856, during the local epidemic of yellow fever, which is still remembered, he organized the Relief Society, of which he was President.

After the outbreak of the War for the Union, he entered the United States army, Feb. 27, 1862, as second lieutenant in the 12th N. Y. Volunteers. Promoted soon after to be first lieutenant, he commanded his company at Gaines's Mills, June 27, 1862—where he was wounded—the second Bull Run battle, Aug. 20, 1862; Antietam, Sept. 17, 1862; and Fredericksburg, Dec. 13, 1862. Subsequently, he served on the staff of Gen. Butterfield, then on that of Gen. Meade, and then on that of Gen. Hooker, in the Army of the Cumberland. He received, May 16, 1862, the much-coveted medal of honor from Congress for his gallantry at the battle of Resaca, Ga. In 1864, as chief of staff to Gen. Butterfield, he participated in the battle of Lookout Mountain, and subsequent campaigns near Atlanta. Subsequently transferred, at his own request, to the Army of the Potomac, he bore a responsible and honorable part in the final operations of the war, and was detailed to assist in paroling the soldiers of the Confederacy after the surrender at Appomatox, in 1865. In May of that year, he received his honorable discharge from the United States army, which he left, after long and brilliant service, for which he received a brevet as Brigadier General.

Upon returning to civil life, General Oliver engaged in the coal business with Samuel Bonnell, Jr., his brother-in-law. Removing to Wilkes-Barre in 1869, he built there a small powder-mill. Subsequently, he organized the Luzerne Powder Co., bought out an older concern engaged in the same business, and established the famous mills at Laurel Run, near Wilkes-Barre. The machinery in this plant, invented by him, and the arrangements and rules of operation which he devised, reduced to a minimum the danger of destructive explosions. In 1903, he sold the plant to the Dupont Powder Co., and, retiring from business, resided at Oliver's Mills for the rest of his life.

General Oliver became an Associate of the Institute at its first meeting, held at Wilkes-Barre, in May, 1871. In 1877, a second Wilkes-Barre meeting was held, in connection with which he entertained the Institute party at Laurel Run, where opportunity was given for the inspection of his process in the manufacture of gunpowder (see *Trans.*, vi., 5). In October, 1891, during the Glen Summit meeting, a picturesque and unique "camp-fire" reception, given by him in the beautiful forest near his residence, was attended by hundreds of members and guests, who lingered until midnight, enjoying the ingenious and charming entertainment provided by this veteran soldier for his comrades in the campaigns of peaceful industry. At this gathering, an interesting report from a Special Committee, appointed at a previous session of the Institute, was adopted by acclamation. (This report, for which see *Trans.*, xx., lxiv., recalls the events and leaders of the Institute for the past 20 years, and cordially recognizes the generous co-operation of General Oliver).

Among other societies of which he was a member, were the Military Order of the Loyal Legion, the Society of the Army of the Potomac, the Huguenot Society, the Society of the War of 1812, and the Netherland Society of Philadelphia—a list which indicates both his honorable ancestry and his own distinguished career.

At the age of 81, after many years of exposure to the perils of war, and of the most dangerous business known outside of actual warfare, General Oliver received from an accidental fall cerebral injuries which terminated in his death, May 18, 1912, at his own home at Oliver's Mills. It is said that he had almost never been ill, and that, up to the last, he was accustomed to go often to Wilkes-Barre, and attend to business. His sudden and painless death, delivering him from the slow decay of age, must be deemed a crowning blessing upon his long and illustrious life. R. W. R.

W. J. Sutherland was born in January, 1844, at L'Avenir in the province of Quebec, and educated at the Academy of Ulverton, Quebec, and at McGill College, Montreal. Removing to the State of Maine, he taught school there for a time, and later served, as a member of a Maine regiment, in the war for the Union. After the close of that war, he went to Cuba, where he distinguished himself in the Spanish service, was severely wounded, and received the rank of Colonel. Returning to the United States, he worked for a while under the Coast and Hydrographic Survey, then studied law in Boston, and at the Albany law-school, and was admitted to practice in 1875.

In 1879 Colonel Sutherland became interested in mines at Candelaria, Nev., and, increasing his holdings from time to time, also secured the aid of foreign capital, built a pipe-line bringing water to Candelaria from the White mountains, and erected a large silver-mill, which continued in operation until the great decline in the price of silver rendered it unprofitable. About 1900, Mr. Sutherland undertook the development of what is now known as the profitable Alaska-Perseverance gold-mine, near Juneau, Alaska. At the time of his death, which occurred April 22, 1911, after a brief illness, at his home in Twickenham, Eng., he was still connected with this enterprise, in which he was a large stockholder.

Mr. Sutherland was a life-member of the Institute, having been elected in 1899.

SUBJECT TO REVISION.

[TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS.]

Electrical Fume-Precipitation.

BY F. G. COTTRELL, SAN FRANCISCO, CAL.

(New York Meeting, February, 1912.)

About a year and a half ago, at the San Francisco meeting of the American Chemical Society, in connection with the excursions to local smelting-works, I had occasion to show some lantern-slides illustrating the recent commercial development of electrical methods for precipitating the suspended matter from smelter-smoke, the underlying phenomena of which were clearly brought to the attention of the technical public by Sir Oliver Lodge as long ago as 1884. In the *Journal of Industrial and Engineering Chemistry*, August, 1911, most of these views were reproduced, together with an account of the work from its beginning up to July, 1911.

To-day I wish particularly to show you some of the further developments of the work since that time; but in order to make these clear they will be prefaced by a brief abstract of the article above referred to.

The removal of suspended particles, from gases, by the aid of electric discharges is by no means a new idea. As early as 1824 we find it suggested by Hohlfeld[1] as a means of suppressing ordinary smoke, and again a quarter of a century later by Guitard.[2] These suggestions, which do not seem to have stimulated any practical study of the question, were soon entirely forgotten, and only brought to light again by Sir Oliver Lodge,[3] many years after he himself had independently rediscovered the same phenomena and brought them to public attention[4] in a lecture before the Liverpool Section of the Society of Chemical Industry, Nov. 3, 1886. The first recorded attempt to apply these principles commercially appears to have been made at the Dee Bank Lead Works. The general principle of electrical precipitation of suspended matter was at this time patented by Alfred O. Walker, of the above firm, in several countries,[5] but these patents have long since expired. The apparatus was installed in 1885 by the works-manager, W. M. Hutchings, with the co-operation of Professor Lodge, and briefly described by the former,[6] just before its completion, as consisting of a system of metallic points situated in the flue from the lead-furnaces, and excited from two Wimshurst influence-machines, with glass plates 5 ft. in diameter, each machine being driven by a 1-h-p. steam-engine.

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[1] "Das Niederschlagen des Rauchs durch Electricität Hohlfeld," *Kastner Archiv Naturlehre*, vol. ii., pp. 205 to 206 (1824).

[2] C. F. Guitard, *Mechanics Magazine* (Nov., 1850).

[3] Historical Note on Dust Electrification and Heat, O. J. Lodge, *Nature*, vol. lxxi., p. 582 (1905).

[4] The Electrical Deposition of Dust and Smoke, with Special Reference to the Collection of Metallic Fume, and to a Possible Purification of the Atmosphere, *Journal of the Society of Chemical Industry*, vol. v., No. 11, pp. 572 to 576, with appended bibliography (Nov. 29, 1886).

The apparatus undoubtedly did not in practice fulfill expectations, as we find nothing further of it in the literature. The most apparent weakness of the project lay, perhaps, in the reliance on the Wimshurst machine, which had then just been brought out, and from which a great deal more was anticipated[7] than has been justified by experience, at least as far as commercial applications are concerned.

None of these suggestions and patents had been successful in commercial operation when, about 1906, occasion was taken to repeat Lodge's early experiments, and to endeavor to reduce to engineering practice, under more favorable conditions, the principles and processes developed by Lodge and Walker.

The precipitation of suspended matter in gases or liquids may be accelerated by either the alternating or the direct electric current. The former acts chiefly by agglomerating the suspended particles so that, constituting larger bodies, they settle by gravity more rapidly. Such an application has been made in Europe, it is claimed, by sending powerful Hertzian waves into foggy air. Another is the use of the alternating current devised by Buchner Speed and me for separating emulsified water from crude California petroleum.[8]

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[5] Great Britain, 11,120, Aug. 9, 1884; Belgium, 68,927, May 19, 1885; Spain, 7,211, July 10, 1885; Germany, 32,861, Feb. 27, 1885; Italy, 18,007, Mar. 31, 1885; United States, 342,548, May 25, 1886.

[6] *Berg- und Hüttenmännische Zeitung*, vol. xliv., No. 25, pp. 253 to 254 (June 19, 1885).

[7] A. O. Walker, *Engineering* (London), vol. xxxix, pp. 627 to 628 (June 5, 1885). G. Tissandier, *Electrician* (London), vol. xvii., p. 33 (May 21, 1886).

But for the voluminous gases of smelters, moving rapidly through flues—or even expanded into dust-chambers—this agglomerating and settling process is too slow; and recourse must be had to the direct current. If a needle-point, connected to one side of a high-potential direct-current line, be brought opposite to a plate connected to the other side of the line, the space between, and any insulated body contained in it, become highly charged with electricity of the same sign as the needle, whether positive or negative; and such a body, if free to move, will be attracted to the plate of opposite sign. Suspended particles of fume may thus be precipitated, not slowly by gravity, but instantly by electricity, upon electrodes.

To make this action commercially practicable, the first step was to get an economic and effective source of high-tension direct current. In the installations here described, this was done by transforming an ordinary alternating current up to 20,000 or 30,000 volts, and then commutating to an intermittent direct current, by means of a rotary contact-maker driven by a synchronous motor. This high-tension direct current is applied to a system of electrodes in the flue carrying the gases to be treated.

It has been largely in the development of practical forms and arrangements of electrodes to meet the exacting and varied conditions of large-scale operations in different installations that the most difficult as well as the most interesting problems have presented themselves and have gradually been worked out to successful solutions.

The first experiment on a working-scale was made at the Hercules powder-works at Pinole, Cal. The gases treated contained about 4 per cent. by volume of dry, gaseous sulphur trioxide, and were brought into contact with water, which, combining with the sulphur trioxide, formed the far less volatile sulphuric acid, which immediately separated as a dense white cloud of suspended particles, so fine as to constitute one of the most difficult of all materials to remove by filtration, but easily and successfully precipitated by the electric current.

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[8] Dehydration of Crude Petroleum, a New Electrical Process, by Arthur T. Beazley, *The Oil Age*, vol. iii., No. 13, p. 2 (April 21, 1911). Water Emulsions in Crude Petroleum, by Allen C. Wright, *Western Engineering*, vol. i., No. 3, pp. 207 to 210 (June, 1912).

The next installation was made at the Vallejo Junction works of the Selby Smelting & Lead Co., where the stack of the refinery discharged the mists which escaped from the boiling sulphuric acid used to dissolve silver. These were successfully treated, and the electrical precipitation process has been in steady commercial operation there ever since.

The problem next attacked was the treatment of the gases from the pyrites- and matte-roasters of the same plant. Here the material to be removed was a mixture of solid dust and fume, with liquid sulphuric acid. The latter often exceeded in weight all the solids in the gas. Moreover, the gases were highly charged with water-vapor, which diluted the acid and produced a muddy deposit. These features required a new construction, by which the gases were treated in flues of sheet-lead.

A later and larger installation was made at the Balaklala smelter at Coram, Shasta county, Cal., where more than 250,000 cu. ft. of gas per minute was treated.

The work at Balaklala may fairly be likened to one of those "successful" surgical operations where the patient incidentally dies. The process worked, and technical difficulties were one after another overcome and efficiency steadily increased during the months of operation up to the very end, but general conditions did not justify the continued operation of the smelter, and it finally closed.

The installation served, however, to thoroughly demonstrate that there was nothing fundamentally impracticable about applying the method to plants of any desired size. In the paper in the *Journal of Industrial and Chemical Engineering*, of which the foregoing is a brief and partial abstract, details (including drawings and photograph) of the various plants may be found.

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After the Balaklala, the next large problem to be actively undertaken was the equipment of the plant of the Riverside Portland Cement Co., in southern California. The orange-growers of the surrounding district had complained of the dust thrown out from the rotary kilns, and this company was very desirous of abating the nuisance.

A general view of this plant is shown in Fig. 1. The kiln-stacks are seen just protruding above the roof of the kiln-room. On top of this building and exactly in the middle of the picture is the initial experimental installation of two small precipitating-units, built on practically the same lines as those at the Balaklala plant, but of very much smaller capacity. With the help of these, the new conditions, such as temperature and character of dust and gases, were first studied, and then construction was commenced on a full-sized treater to handle the 50,000 cu. ft. of gases per minute from one of the ten kilns in this mill. This is the structure seen at the right or opposite end of the row of stacks.

The effect on the appearance of the issuing gases due to turning on the electric current, the solid particles being all retained within the treater, is shown in Fig. 2. These two views were taken only a few moments apart under otherwise identical conditions. The smoke of the other nine kilns is, of course, in no wise affected. Fig. 3 is a closer view of this same treater.

The work of similarly equipping the other nine kilns is now under way, and by the end of the year it is hoped to have the entire plant thus equipped.

An interesting development of this installation has been the finding in the collected dust of considerable amounts of potash-salts soluble in water, and the possibility which this suggests of definitely seeking high-potash raw materials for cement-manufacture, for the sake of the by-products.

The effect of turning on the current in a small experimental precipitator handling about 3,000 cu. ft. of gases per minute coming from basic-lined copper-converters blowing a somewhat leady matte, at the Garfield plant, is shown in Figs. 4 and 5. The collecting-electrodes in this case are tubular in form with the discharge-electrode in the axes of the tube. A full-scale working-unit, capable of treating all the gases from one large basic converter, is now under construction at this plant, and will probably be completed by June, 1912. In this plant the sole object is the recovery of values, as fume-nuisance problems have not arisen.

Perhaps not the least interesting phase of this work centers around an experiment in economics to which the larger portion of the patent-rights covering the process in this country are being devoted.

As mentioned in the introduction, this work grew from experiments started in the Chemistry Department of the University of California. When it became necessary to extend these to operations on a commercial scale, it was three alumni of that department, E. S. Heller, Dr. Harry East Miller, and Prof. Edmond O'Neill, who furnished the sinews of war and helped to guide the business administration, under an informal mutual understanding that, as soon as the venture repaid the investment with reasonable interest, a considerable portion at least of the remaining patent-rights should be turned over either to the University or to some other academic institution, as the nucleus of an endowment for scientific research.

By the time the business had reached this degree of financial success, it was no longer local in its application; and it was felt that it should be in the hands of a more widely representative institution than a State University. Consultation with the officers of several of our national scientific and engineering societies made it clear that none of these societies was organized to administer adequately such a business; but finally, through Director Holmes, of the Bureau of Mines, the matter was brought to the attention of the Smithsonian Institution, and a formal offer of the patent-rights still remaining unsold was made to that body.

In consultation with the Board of Regents of the Institution it was finally decided most expedient to create a separate board of administration to manage business matters in connection with licensing and further development of the patents, and turn over the net profits to the Smithsonian or other scientific and educational institutions or societies for use in scientific research. This Board has recently been organized under the name of the "Research Corporation," with offices at 63 Wall Street, New York City, and will be conducted like any other business corporation, save that no dividends are paid to stockholders, the whole of the net profits being devoted to scientific research.

Already a number of other patents are being offered to the corporation, both from this country and abroad, for the move-

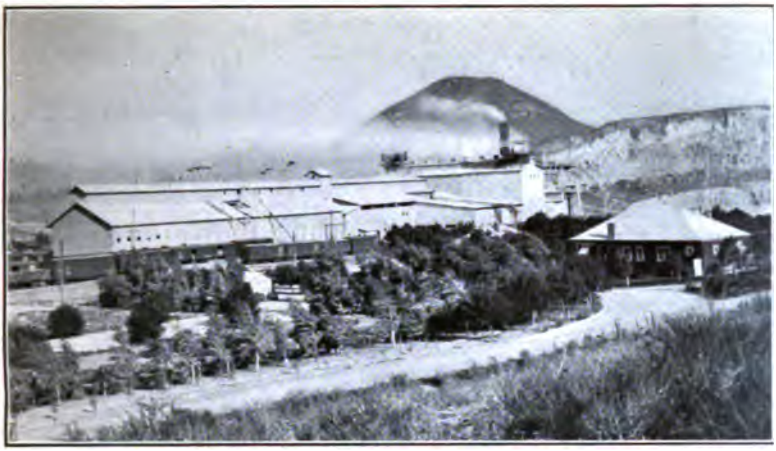

FIG. 1.—PLANT OF RIVERSIDE PORTLAND CEMENT CO., SHOWING GAS-TREAT ON ROOF. CURRENT OFF.

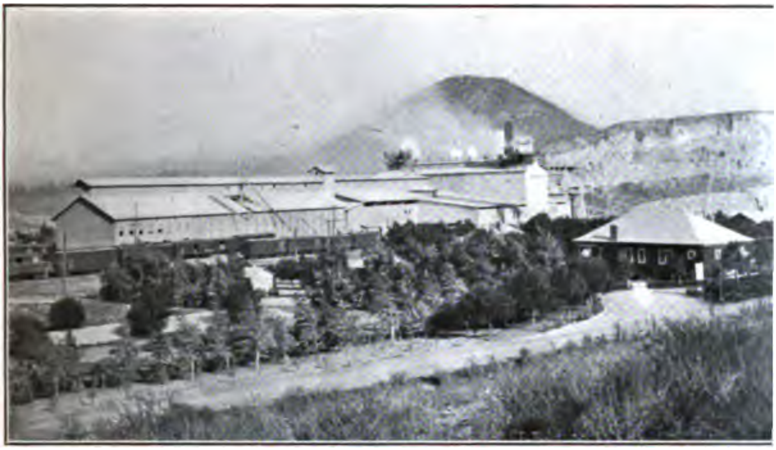

FIG. 2.—PLANT OF RIVERSIDE PORTLAND CEMENT CO., SHOWING SUPPRESSI OF SMOKE AFTER TURNING ON CURRENT.

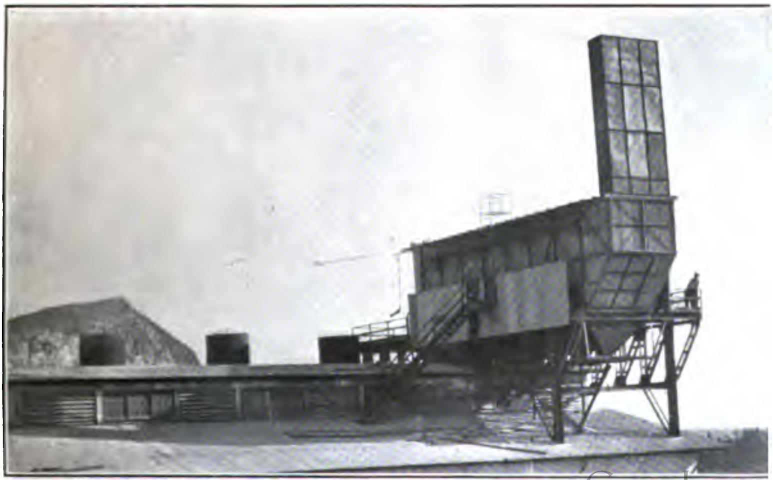

FIG. 3.—GAS-TREATER AT PLANT OF RIVERSIDE PORTLAND CEMENT CO.

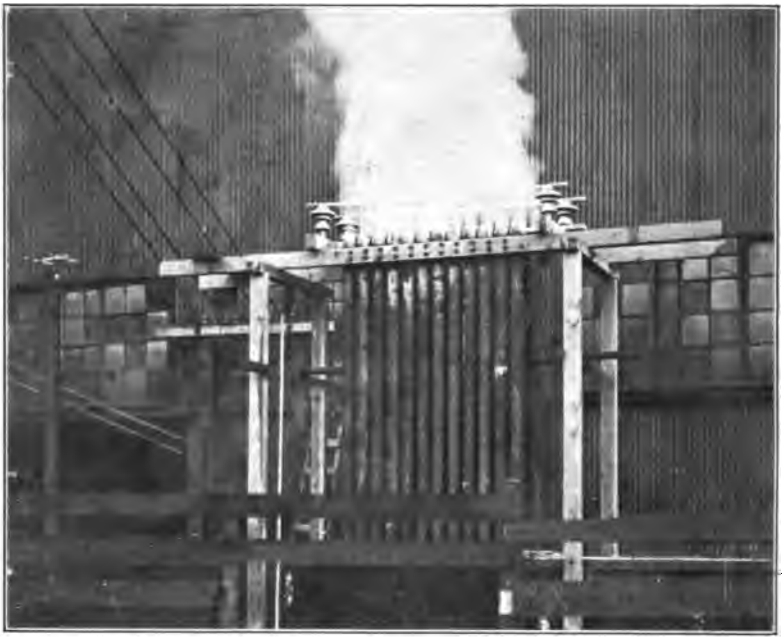

FIG. 4.—EXPERIMENTAL TUBULAR TREATER AT THE GARFIELD PLANT BEFORE APPLICATION OF CURRENT.

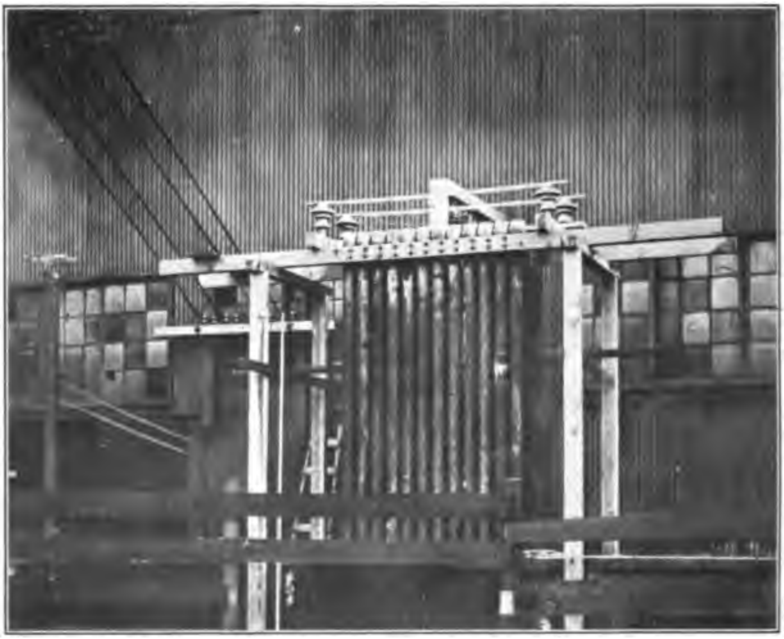

FIG. 5.—TUBULAR TREATER WITH CURRENT ON.

ment has no geographical limits, either as regards sources of revenue or expenditure of income.

As the corporation will soon issue detailed literature descriptive of its aims, objects, and methods, it will be needless to detain you here with these, but your interest and encouragement are earnestly bespoken in behalf of this new project.

DISCUSSION.

F. W. C. SCHNIEWIND, New York, N. Y.: Have you any idea as to cost of operation?

MR. COTTRELL: I omitted to speak of the cost of installation. The precipitation-plant at the Balaklala smelter, up to the time of initial operation, including the construction of flues, installation of the apparatus, etc., cost $107,000, and subsequent modifications probably represented at least $15,000 more. Upon the information gained in this plant, the installation at the cement-mill, handling nearly twice as much gas, has been reduced in cost in a number of ways, but the overhead construction, carrying the precipitator at the top of the stacks, is naturally expensive, yet, even so, the total cost of construction will probably not greatly exceed $150,000.

The mechanical details of the tubular form last shown have been worked out more recently than those of the plate form, and the tubular form promises to be still less expensive in construction, as you would naturally expect. The cost of operating the plant varies somewhat with the character of the gases used. The amount of power consumed is not determined by the tonnage of the material collected, but by the volume of gas to be handled. It takes as much power to precipitate a thin cloud as a dense one.

In the cement-plant they are getting a higher efficiency; *i. e.*, a greater number of cubic feet treated per kilowatt than in the earlier plants. This is perhaps partly due to higher temperature of the gases, which here average about 450° C. The best figures on the cement-plant are between 3 kw. and 4 kw. of electrical energy for the unit shown in the picture, which is an entirely insignificant quantity of energy for the amount of gas treated, representing for the complete installation probably less than 1 per cent. of the power used for other purposes about the mill. The new units being larger, however, will consume somewhat more energy.

Dr. Schniewind: What is the tonnage production?

Mr. Cottrell: The mill is rated at 4,000 barrels of cement per day. Smaller installations would naturally consume somewhat more energy per cubic foot of gas treated. In the Balaklala plant, the power-consumption was about 120 kw. This higher power-consumption was partly due to the less perfect state of the art and partly to different conditions of the gases under treatment.

E. Gybbon Spilsbury, New York, N. Y.: I wish to ask about the coating of the plates with deposit; first of all, how this is thrown down, and also whether, after long use, there is not a gradual building up, hardening, and attaching to the plates themselves, of a coating which might gradually choke up the whole thing, and cause those products, which are deposited at so great a heat that they would almost fuse together, to be an obstruction in the operation of the plant. In the flues there would be an aggregation of flue-dust which would gradually cement together.

Mr. Cottrell: That is one thing that has to be met very carefully in the engineering of the plant by determining the proper temperatures at which to install the apparatus. The plants thus far installed have furnished an agreeable surprise by presenting less of these troubles than was expected. The first plant treated simply sulphuric acid fumes, which became liquid and ran off the plates. The next was in connection with roaster-gases at the Selby plant, where they were roasting a mixture of California concentrates. The fumes here were a mixture of sulphuric acid and flue-dust, with some arsenic and lead sulphate, and it was difficult to keep the apparatus from caking up. The work began with the idea of putting in iron plates in a brick flue. After making careful tests, it was decided that, in order to accomplish what was wanted in the way of suppressing all the fumes, it would be necessary to cool the gases sufficiently to insure all the sulphuric acid being condensed from the gaseous state into liquid droplets. This made the precipitated mixture so muddy that it was necessary to abandon the iron and brick construction and substitute lead flues and lead electrodes in its place. In this case the material came down as a mud more or less caky. This necessitated the intermittent operation of individual precipitation-units. Each was operated about 6 or 8 hr. at a time, and then it was necessary to open the top of the flue on hinges like a box and flush the electrodes down with a hose.

It was found that as long as the materials thrown down did not carry more than 10 or 15 per cent. of moisture, the deposit was so spongy that it could be shaken off; but when, for example, the material contained equal proportions of sulphuric acid, moisture, and solids it became necessary to wash it off with the hose.

In the work at the Balaklala smelter variable conditions were encountered. It was at first thought there would be considerable corrosion of the iron flues and electrodes; but after six months' running no serious corrosion was apparent. The apparatus stood extremely rough handling.

In the cement-mills the conditions are again very different, there being no acid or moisture to fear. The conditions probably vary very much in different mills, but, in the Riverside mill at least, the dust comes down in very easy form to handle. They burn a raw mixture, which gives a slight excess of lime in the dust. When dusts with higher alkalies and higher alumina are dealt with, it may be necessary to study the conditions of temperature and composition of the dust more carefully to prevent accretions.

The converter-gas problem, at present, is purely a matter of saving values. There has been a good deal of difficulty in past years in finding methods to collect these particular fumes, which are often very rich in lead as well as other values. The use of the electric precipitation-method in conjunction with the basic-lined converter gives good promise of opening an outlet for many siliceous, low-grade lead-ores which are at present worthless, because they cannot be profitably treated in the lead blast-furnace.

Mr. Spilsbury: What is your method of shaking the plates?

Mr. Cottrell: Different forms of rattling- or hammering-arrangements have been employed to suit varying types of construction, a relatively slight jarring of the plates or tubes being usually sufficient, the dry deposits shaking loose very readily.

Fortunately the work is now getting over on to the ground where it is being done from an economical stand-point rather than from the stand-point of possible litigation. I do not suppose, however, it would have been developed to the extent it has been, except from the pressure which was brought upon it to avoid litigation. It has gone beyond that point now, and is being developed in a much more methodical way on a commercial basis.

A. H. Elliott, New York, N. Y.: In your main talk you said you were using oil for burning the cement. What would happen if the oil was in excess and you got smoky fumes—would the tarry matter prevent the action?

Mr. Cottrell: The soot is precipitated as well as the dust. They are putting an installation into one of the gas-plants in California to treat the smoky residuum from making illuminating-gas from crude oil. There is quite a field of application in that direction. Each one of these applications requires a certain amount of engineering adaptation, but the apparatus has proved itself flexible enough, with proper study of the details of application, to meet all the conditions.

Dr. Elliott: My idea was, that you would want to keep the apparatus clean. We have to consider also the hydrocarbon ingredients in the fumes from the gas of soft coal or oil fires. We have a number of oil-firing apparatus and the smoke is a great nuisance. The thought I had in mind is that it would be an insulator that would be very sticky to remove.

Mr. Cottrell: I imagine in handling a tarry liquid mass you would have some difficulty, and you might have to get over that by special means. The gas-works installation mentioned collects what is practically a clean lampblack or soft carbon.

I used to illustrate this process in a laboratory-demonstration by simply rigging up a tobacco-pipe and blowing the smoke into a glass precipitation-chamber, turning on the current and precipitating it in that form.

I always decry, however, the tendency to hold this system up as a general means of suppressing the coal-smoke nuisance. That should be done, in most cases, by proper combustion in the first instance. When you have good combustion you get rid of the smoke-nuisance. Still, there are isolated special cases where it may be more expedient to electrically precipitate smoke than to consume it.

L. C. GRATON, New York, N. Y.: I think, perhaps, the installation of Mr. Cottrell's which has received most attention from metallurgists is that at Balaklala. The fact that the courts closed that plant, after a trial of the process, has led some people to conclude that it was not satisfactory. From acquaintance with conditions in Shasta county, I think I can say something that possibly Mr. Cottrell did not feel free to say, and that is, that the so-called farmers of that section have for many years carried on a campaign of extortion against the smelters, and there is very little doubt in the minds of disinterested parties that, if the question had been settled on its merits, instead of by biased judges, the Balaklala plant would be operating to-day with Mr. Cottrell's system.

There is one question I would like to ask, not from a metallurgical stand-point so much as indicating the possible presence of certain constituents in the ore, and that is, whether such a thing as bismuth or arsenic was found in the fume?

MR. COTTRELL: That ore seems remarkably free from those heavy metal constituents. As you go up the "Copper Crescent," in Shasta county, there seems to be an increasing quantity of lead, arsenic, gold, and silver. I cannot cite, however, any detailed determinations. I have never seen any statement of the quantity of bismuth present. The lead at Balaklala was extremely low, but I cannot speak with any degree of certainty of the average runs there. I was perfectly willing to keep out of anything that might draw me into the controvery as to what the deleterious constituents were. I was at Balaklala in the capacity of putting in a particular, definite technical device, and I wanted particularly to keep that separate and free from any question of fume-litigation, as far as I was concerned. For that reason, I did not take any special pains to inform myself on the constituents of the material, outside of those matters we had to deal with in our actual operations, but my impression was that the lead, arsenic, and bismuth were surprisingly low in the fume, compared with many other places.

Dr. Schniewind: Has this method ever been employed for the removal of dust from iron blast-furnace gas for use with producer-gas in gas engines?

Mr. Cottrell: That application of it has just begun. Attention was at first centered on the most urgent questions in the West, where the apparatus was originated and developed. I did take up some of that work later, however, and discussed it with some of the representatives of the iron firms in Indianapolis and Chicago; and we had some tentative negotiations under way to take up work along that line, when the relations with the Smithsonian Institution developed, and made it inexpedient to get that work under way under the old management, and, perhaps later, have to make new arrangements; so it was thought better to defer the matter until the new plans were in definite shape. Somewhat similar work is, however, developing in the West in connection with producer-gas and oil-gas engines.

[TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS.]

The Solubility in Nitric Acid of Gold Contained in Certain Copper-Alloys (Copper-Bullions).

BY EDWARD KELLER, PERTH AMBOY, N. J.

(New York Meeting, February, 1912.)

IN a paper, entitled A Uniform Method for the Assay of Copper Material for Gold and Silver,[1] A. R. Ledoux invited the assayers of this country to contribute to a symposium, in which the results of their assays of given samples of copper-matte and metallic copper would be made comparable. This symposium[2] appeared about one year thereafter; and through it and its discussion, first became generally known the fact that the gold-yield by the all-fire method of assaying is usually higher than by the so-called combination (wet and dry) method, in which nitric acid is employed as a solvent for the copper. In the publication alluded to, the supposed cause of gold-losses in the latter method was declared to be the solvent action of copper nitrate, or of the reduction-products of the nitric acid, upon the gold. Up to a very recent date, gold has been held to be insoluble in pure nitric acid. (The solvent action of any chlorine present in the nitric acid must here properly be disregarded as belonging in the realm of carelessness.

Among later allusions to this topic, W. R. Van Liew[3] ascribes the solution of the gold in the combination-assay entirely to the action of nitrous acid at the high reaction-temperature of copper and nitric acid; F. B. Flinn[4] believes the low result in gold to be due exclusively to mechanical losses—finely-divided gold penetrating through the filter-paper—but presents no experimental data confirmative of this belief; and O. Pufahl[5] attributes the solubility of the gold in some coppers to the presence of selenium—a view which appears to be shared by many assayers. There seems, however, to be no foundation in fact for such an assumption. We do know that gold is dissolved by concentrated selenic acid when heated. When copper with minute selenium-content is dissolved in nitric acid, selenious acid is formed, which is highly diluted, and has, to our knowledge, no action whatever on the gold. Recently, F. P. Dewey[6] has convincingly demonstrated that pure gold dissolves in small quantity, on long-continued boiling, in pure concentrated nitric acid.

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[1] *Trans.*, xxiv., 575 (1894).

[2] *Trans.*, xxv., 250 (1895).

[3] *Engineering and Mining Journal*, vol. lxix., No. 16, p. 469 (Apr. 21, 1900).

[4] *Engineering and Mining Journal*, vol. lxxxvii., No. 11, p. 569 (Mar. 13, 1909).

[5] Lunge, *Chemische Technische Untersuchungs Methoden*, vol. ii., p. 229 (1900).

Before discussing experiments and theories on this subject, it may be well to state the chemical composition of the various auriferous copper-bullions, and briefly to describe their treatment in commercial sampling and assaying, since in this field lies the practical importance of a thorough knowledge of the problems involved.

In shape, copper-bullion castings are generally known as pigs (ingots), and slabs (plates), the former being thick and the latter thin, as compared to their other dimensions. The first form favors and the second minimizes so-called segregation. The weight of the castings ranges approximately between 200 and 600 lb. They are sampled by drilling through their entire thickness—the holes being placed according to regular templates; and the drillings are brought to the proper fineness by grinding. In several of our large copper-refineries the grinding process is carried to a point where the whole sample will pass a 16- or even a 20-mesh screen. In the writer's practice these samples are separated into coarse and fine parts by means of a 40-mesh screen. It is self-evident that the fine part will contain all such extraneous matter as the oxides, slags, mold-wash and incidental dirt, while the coarse part consists of clean metallic (alloy) particles. Table I. shows differences in metal-values of the fine and coarse parts which demonstrate the necessity of separate assays of the two and the averaging of the results according to the ratio of the parts; or else of making combined assays of the fine and coarse, mixed and weighed in their proper ratio. Some assayers believe they can accomplish this object equally well by reducing the mixed sample to approximately the desired weight by some mechanical cutting-down device.

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[6] *Journal of the American Chemical Society*, vol. xxxii, No. 3, p. 318 (Mar., 1910).

As regards the methods of assaying the copper-bullions for gold, little need be said of the all-fire method. It assumes that all the copper is eliminated by scorification and cupellation. In these two operations, gold-losses occur by slagging and absorption; but, if these be determined and accounted for, operations by this method under varied conditions yield concordant results, which are the most accurate attainable. The nitric-acid combination-method presents a very different case. Here, variation in heat-conditions, or in the dilution of the acid used in dissolving the copper, causes varying quantities of the gold to be dissolved, and therefore lost.

In the tests embodied in this paper, a uniform procedure was employed. To 1 assay-ton of the copper was added 190 cc. of distilled water with 90 cc. of pure nitric acid (sp. gr. 1.42); and after the violence of the reaction had subsided, the beaker was placed on a steam-plate, at 160° F., where it was left until the red fumes had been expelled. No chloride was used to precipitate the silver; but the gold was gathered on a triple S. & S., No. 597, 12.5 cm. filter covered with 2 g. of test-lead; and, after incineration of the filter and the addition of 20 g. of test-lead, a sufficient quantity of silver, in the form of nitrate crystals, was added. That part of the combination-method designated as "dry" needs no description, since what has been said of the all-fire method holds true here, namely, that when the so-called fire-losses are determined and accounted for, variations in this part of the method do not alter the results.

When, therefore, we have determined the amount of gold by the all-fire method and the amount by the combination-method, the difference should be equal to the gold-loss by solution in the latter method. (The suggestion that this difference may involve other factors will be discussed hereafter.) All the results for gold given in this paper are averages, generally of five assays, often of more. The accuracy of the all-fire method is, almost without exception, within 0.03 oz. per ton, while the combination-method sometimes shows on the higher-grade bullions differences as high as 0.2 oz. per ton. The measurement by the ounce per ton has been employed throughout for silver and gold, not because this is the commercial custom, but because it is much more convenient for comparison than the extended decimal fractions expressing percentages. In carrying out the various assays and analyses, I was assisted by K. W. McComas and Albert Ferrell.

From Table I. it will be seen that, on account of the admixture of extraneous impurities, already referred to, the copper-content in the fines is always lower than in the coarse, while in many cases (class A) the fines show an enrichment in other elements.

The examples of copper-bullions in Table I. have been divided into two classes, A and B. In general, it may be remarked that to class A belong all those whose copper-tenor is above 97 per cent., and to class B those whose copper-tenor is below that figure. In the castings of class A, the precious metals and other elements are found concentrated towards the freezing-center of the castings, while in class B they are found richer towards the surfaces.[7] None of the elements in question, however, show the same degree of heterogeneity, so that, in some instances, the two classes overlap each other. Viewed as solutions, class A must be considered as copper subsaturated above its freezing-point (hypo-eutectic); while in class B the copper is saturated above its freezing-point (hyper-eutectic). In class A the copper is the element freezing first; in class B the other elements freeze first.

Looking again at the composition of the samples, fine and coarse, it will be noted that under class A the fines, generally, are higher in foreign elements than the coarse; and it is consistent to conclude that the fines (exclusive of extraneous matter), in their greater portion, are derived from the later-freezing parts of the bullion castings. The gold in the fines shows a markedly greater solubility in nitric acid than the gold in the coarse, which, as will be seen later, is in conformity with the supposition that the fines are derived from the last and most slowly-cooled portions of the castings. The fines in class B show a smaller amount of impurities than the coarse, thereby also proving their origin from the last-freezing portions of the bullion. In them, moreover, the gold-solubility is much greater than in the coarse.

The samples of bullions, as regards their metallurgical origin, are as follows: Nos. 1 and 2, converted copper, furnace-refined; Nos. 3, 4, 5, 6, and 7, converted copper, gas-finished;

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[7] E. Keller, *Trans.*, xxvii., 106 (1897).

No. 8, reverberatory blister-copper; Nos. 9 and 10, blast-furnace black copper.

My first serious attention to the solubility of gold in nitric acid was drawn by the peculiar facts connected with bullions Nos. 1 and 2, Table I. They were derived from the same source and treated identically in the process of refining; often coming from the same furnaces, which statement finds corroboration in the results of the partial analyses. Both were quenched (pickled) as soon as solidified. Two parallel series of gold-assays were made for many millions of pounds of material; one series being by the all-fire method, the other by the combination-method. The results of these two series of assays were, that for bullion No. 1, the gold-yield by the combination-method showed a deficit of 13.71 per cent., as compared with the all-fire method, while for No. 2 the corresponding figure was 26.62 per cent.—both, it may be repeated, under identical conditions of assaying. Since, as already remarked, all the assays were corrected for the so-called fire-losses, it followed that the results were to be so interpreted that the figures 13.71 per cent. and 26.62 per cent. denoted the solubility-losses of the gold in the two respective bullions—in one nearly twice as great as in the other. Although these two bullions were the same in origin, composition, and metallurgical treatment, there remained one evident difference in the castings, as to size and weight; No. 1 weighing about 300 lb., and No. 2 over 500 lb. This fact implied a difference in the rate of cooling, and led at once to the conclusion that gold contained in slowly-cooled copper is more soluble in nitric acid than that in rapidly-chilled copper. That the gold-solubility was not influenced by any mechanical differences in the two samples is evidenced by the fact that the ratios of fine and coarse in both are very nearly alike, while the size of the coarse particles in both samples is the same, yet the solubility of the gold in the coarse is for No. 1 only one-half of that in No. 2.

If, then, the conclusion reached be true, that the solubility in nitric acid of the gold in the same bullion or alloy is subject to great variation according to the rate of cooling, it should be possible to reduce this solubility to a minimum by the instantaneous chilling of the molten bullion. Granulation, or shotting, by running a stream of the molten bullion into cold water, suggested itself, and advantage was taken of a blast-furnace charge of copper, for an experiment, to compare the behavior of the gold in a shot-sample with that in a drill-sample from a large and slowly-cooled slab. The results of the tests are incorporated in Table I., bullion No. 10. They demonstrated at once the correctness of the hypothesis; none of the gold in the granules or shot being soluble, while the absolute amount of gold from the slowly-cooled slab dissolved by nitric acid was greater than in any other of the samples of copper-bullion or alloys that have been tested.

Thus far, the conclusions reached were based on the analytical methods; it now appeared desirable to proceed to the synthetical.

Table II. shows the solubility of the gold in shot and drillings of a series of alloys, each of which was prepared by adding 0.5 per cent. of a foreign element and an amount of gold equaling approximately 2 oz. per ton to commercial electrolytic copper. The alloy was thoroughly fused and mixed under a charcoal cover in a crucible. About 3,000 g. of the alloy was cast into a heated mold to form a small ingot; the remainder being granulated. Each ingot was drilled in the same manner for a sample, while the granules could directly be used for analysis. It would be venturing too much to claim for the solubility-figures in the table that they are the true relative expressions of the influence of each of the given elements on the solubility of the gold contained in the ingots, for there was, probably, a wide variation between them in the conditions of cooling and freezing. In the process of granulating it is far more likely that those conditions are nearly alike in all cases, and that here the effects of the alloyed elements on the gold-solubility are shown in their proper relative degrees. It seems to be demonstrated that the quenching effect diminishes from the metallic towards the metalloidal end of the series, reaching zero with selenium.

The largest quantity of soluble gold thus far found is 0.495 oz. per ton, or 15.55 per cent. of the total gold in this special case, or 0.0017 per cent. of the total alloy—which is a very small proportion. It became of interest, therefore, to determine if this figure could be materially enlarged by the increase of either the gold- or the impurity-content of the copper-alloy.

Table III. shows the negative results on alloys with varying tenors of gold and arsenic. In these cases small ingots were cast in heated molds, which might not preclude too rapid cooling; therefore, tests of the slowest possible cooling were instituted, one of which is exemplified in Table IV., with negative results. This shows also how the great gold-solubility in the fine portion of a sample is reduced by remelting and granulation.

Since neither gold-increase nor impurity-increase appeared to augment the quantity of soluble gold, it became a question what influence pure copper would have in the solubility of a relatively small gold-content. Table V. gives the results of a test with the purest commercial electrolytic copper to which gold was added by melting. It shows that the gold is as soluble in this as in many of the ordinary bullions; but, since the purest commercial electrolytic copper[8] still contains as much, or more, of arsenic and antimony, per cent., as there is soluble gold in most of the bullions, and absolutely pure copper is not available in quantity, the effect of the latter remains unknown.

The tests on the solubility of gold in nitric acid were also extended to a number of lead-bullions; all with results that closely approached the negative. Table VI. gives those obtained from a comparatively large and slowly-cooled casting.

Reference has already been made in the beginning to the bearing of this subject upon the practical sampling and assaying of auriferous copper-bullions. Ignorance in this field has, in the past, led to some acrimonious controversies, the possibilities of which may be understood most readily through an illustration: A furnace-charge of molten copper would be sampled by shotting (properly done)[9]; then several samples by drilling would be taken from slabs, varying from a thin sample plate to the largest-sized commercial slab. The gold-assays on all the samples would now, by one party, be made by the all-fire method, with concordant results, and with the verdict that all the methods of sampling are correct. A second party, believing the combination-method reliable, would, by its use, find concordant results between the shot-sample and the thin-plate sample, while with the other samples the results would show an increasing deficit, reaching its maximum, perhaps 30 per cent., with the sample from the largest slab. This party's pronouncement would be: shot and thin-plate sampling correct; samples from large casting absolutely wrong.

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[8] E. Keller, *Mineral Industry*, vol. viii. (1899).

[9] On the pitfalls of sampling by shotting see: E. Keller, *Trans.*, xxvii., 106 (1897). Wm. Wraith, *Trans.*, xli., 318 (1911).

It may be observed in passing that the accuracy in the gold-assay has been multiplied many times during the past decade through the remarkable improvement in the construction of gold-balances.

At the present day there probably exist but few contracts in the copper-world which permit the employment of the nitric acid combination-method for the assay of gold in copper-bullions, although this method may still be pronounced quite permissible for quick routine-work, when properly modified, and, especially, when applied to quenched or otherwise rapidly-cooled samples.

Summary.

I. *Observed Facts.*—Gold contained in copper-alloys, or bullions, is soluble in nitric acid in varying degrees; the solubility diminishing with the increased rate of cooling of the castings, and reaching a minimum when the molten bullion is quenched in cold water. In nearly all of the quenched samples the gold, determined by the combination-method of assaying, still shows a deficit as compared with the amount found by the all-fire assay; the maximum of this deficit being approximately 2 per cent.

The highest amount of soluble gold was found to be 0.0017 per cent. of the total bullion; no increase in gold- or impurity-content was able to augment that amount.

A definite relation between gold-solubility and impurity-content of the bullions has not been established; the former being more dependent on the rate of cooling than on the quality or quantity of impurity. This proposition is demonstrated by the fact that the gold-solubility in all samples is greatest in the fine portions, which in classes A and B are the most slowly-cooled parts (this conclusion being deduced from their chemical composition). In class A the fine portion is more impure than the coarse; in class B this relation is reversed.

The observations by Dewey of the solubility of pure gold in boiling concentrated nitric acid, and those of Van Liew for avoiding the solution of gold in copper-bullion by dissolving the latter in very dilute nitric acid under ice-cooling, are not controverted.

II. *Hypothetical Conclusions.*—The smallness of the absolute amount of soluble gold contained in bullions, makes it doubtful whether the actual condition of this gold can, at present, be determined by any of the known methods, chemical or physical.

From what is known of the action of strong nitric acid, or other chemicals, on pure or uncombined gold, it would follow that these would not discriminate between such gold obtained in slowly-cooled and in quenched bullions. It is, furthermore, not probable that nitric acid, of the dilution used in dissolving these bullions, and in the presence of a great excess of metallic copper, would attack uncombined gold. The difference in the properties of the gold in slowly-cooled and quenched bullions will, therefore, rationally be sought in a change of chemical conditions, or of the chemical nature of the alloy itself, by the heat-treatment. It seems justifiable to assume that a portion of the gold in slowly-cooled bullion combines with some, or all, of the accompanying elements, forming compounds soluble in nitric acid; that the reaction takes place through a narrow range of temperature above the freezing-point, and that it is a slow one as compared to the speed of cooling; that at higher temperature the gold-compounds dissociate; and that quenching of the bullions at those temperatures leaves the gold in the uncombined state, and, therefore, insoluble in the nitric acid used to dissolve the bullion. Since neither gold- nor impurity-increase augments the amount of soluble gold beyond a low maximum, it would appear that the soluble gold-compounds are formed only in very dilute solutions, or under low osmotic pressure.

The fact that with the seleniferous alloy quenching has no reducing effect on the gold-solubility, may be explained by the supposition that probably the gold-selenium compound forms at higher temperatures than the others, or that it has not dissociated at the temperatures obtaining in the furnace or crucible at which quenching is performed.

The 2 per cent. or less of gold-deficit, as found by the combination-assay in quenched bullions, may be due to a residual soluble gold-compound, formed on account of the impossibility of instantaneous chilling by quenching. Should this explanation be unacceptable, then it could be conceded that this loss is due to all other causes that have been suggested for the loss of uncombined gold.

Phenomena of similar nature to those suggested in the foregoing have been well established by scientific investigation. C. T. Heycock and F. H. Neville[10] have demonstrated that gold in metallic solution forms chemical compounds with metals, as for instance, tin as a solvent, gold, and cadmium combine in the alloy to form in succession the following compounds: Au_4Cd_4, Au_5Cd_5, Au_9Cd_9. One of the instances best known to metallurgists of a chemical compound being formed at a certain temperature and being dissociated at higher ones, is the case of iron carbide (cementite) in steel. The carbide is formed when steel is slowly cooled, and its formation is prevented when the steel is quenched above the dissociation-temperature; chemical conditions are here changed by heat-treatment. In other respects there is much dissimilarity between the carbide in steel and the hypothetical gold-compounds in copper-bullion. The carbide is formed well below the freezing-point of the steel; it is not attacked by dilute sulphuric and hydrochloric acids, while in quenched steel the dissociation-products readily combine; the iron to form a salt, the carbon a hydride. In the quenching process the heat of combination and crystallization remained latent and this becomes instrumental in forming the compound with the acids. An equally well-known case in which, by heat-treatment, a physical change only is wrought, is that of the quenched blast-furnace slags. When these are permitted to cool slowly they attain the crystalline state and become insoluble in acids, while when the molten slags are run into cold water they remain amorphous; the heat of crystallization remains latent, and they are readily attacked by the acids.

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[10] *Journal of the Chemical Society* (London), vol. lix., p. 936 (1891).

TABLE I.—*Copper-Bullions: Character; Weight of Castings; Partial Analyses; Gold-Solubility in Nitric Acid.*

Class.	Remarks.	Samples.	Ratio. Fine/Coarse.	Cu.	Sb.	As.	Ag.	Gold.				Gold-Solubility.
								Assay-Method.		Difference.		
								Fire.	Comb.			
				Per Cent.	Per Cent.	Per Cent.	Oz. Per Ton.	Oz. Per Ton.	Oz. Per Ton.	Oz. Per Ton.		Per Cent.
A.	No. 1. Converter; Refined; 300 lb.	Fine........	$\frac{1}{20.07}$	98.642	0.0349	0.0392	81.01	0.415	0.218	0.197		47.47
		Coarse.....		99.307	0.0254	0.0395	81.22	0.421	0.370	0.051		12.11
		Average,..		99.276	0.0259	0.0395	81.21	0.4207	0.363	0.0577		13.71
	No. 2. Converter; Refined; 500 lb.	Fine........	$\frac{1}{17.11}$	98.654	0.0359	0.0489	80.45	0.415	0.202	0.213		51.32
		Coarse.....		99.346	0.0261	0.0474	80.13	0.401	0.300	0.101		25.19
		Average..		99.308	0.0266	0.0475	80.14	0.402	0.295	0 107		26.62
	No. 3. Converter; Gas finish; 225 lb.	Fine..	$\frac{1}{14.26}$	98.130	0.0531	0.0542	45.30	0.292	0.124	0.168		57.53
		Coarse		99.342	0.0483	0.0434	41.90	0.278	0.246	0.032		11.51
		Average..		99.263	0.0486	0.0441	42.14	0.279	0.238	0.041		14.70
	No. 4. Converter; Gas finish; 230 lb.	Fine	$\frac{1}{23.52}$	93.139	0.0610	0.0743	95.13	0.618	0.379	0.239		38.67
		Coarse.....		98.860	0.0448	0.0219	87.52	0.678	0.633	0.045		6.64
		Average..		98.627	0.0455	0.0240	87.83	0.676	0.623	0.053		7.84
	No. 5. Converter; Gas finish; 205 lb.	Fine........	$\frac{1}{8\ 50}$	95.756	0.0289	0.0338	64.67	0.0707	0 0240	0.0467		66.05
		Coarse,....		98.896	0.0305	0.0220	63.26	0.0707	0.0420	0.0287		40.59
		Average..		98.566	0.0303	0.0232	63.41	0.0707	0.0401	0.0306		43.28
	No. 6. Converter; Gas finish; 230 lb.	Fine........	$\frac{1}{10.25}$	96.046	0.0079	0.0077	69.32	2.264	1.578	0.686		30.30
		Coarse.....		98.858	0.0111	0.0036	66.89	2.274	2 178	0.096		4.22
		Averge ...		98.608	0.0108	0.0040	67.11	2.273	2.125	0.148		6.51
	No. 7. Converter; Gas finish; 400 lb.	Fine........	$\frac{1}{3.149}$	98.350	....	0.2120	88.71	22.821	22.7798	Within the limits of error.		
		Coarse.....		98.742	....	0.2069	83 39	22.1208	22.1615			
		Average..		98.648	....	0.2081	.84.67	22.2897	22.3105			
	No. 8. Reverberat. Blister	Fine........	$\frac{1}{11.50}$	96.432	0.0186	0.0071	60.73	3.992	3.315	0.677		16.96
		Coarse.....		99.120	0.0138	0.0076	55.32	4.275	4.047	0.228		5,38
		Average..		98.905	0.0142	0.0070	55.75	4.252	3.988	0.264		6.21
B.	No. 9. Blast-furnace; Black.....	Fine........	$\frac{1}{9.89}$	95.107	0.0379	0.0776	11.04	0.403	0.174	0.229		56.82
		Coarse.....		97.407	0.0568	0.1225	11.07	0.385	0.364	0.021		5.45
		Average..		97.196	0.0551	0.1184	11.067	0.387	0.347	0.040		10.34
	No. 10. Blast-furnace; single slab 1.5 in. from edge	Fine........	$\frac{1}{2.84}$	89.757	1.358	0.981	176.10	3.105	2.327	0.778		25.06
		Coarse.....		89.980	0.932	1 005	177.80	3.170	2.797	0.373		11.77
		Average..		89.922	1.043	0.999	177.36	3.153	2.675	0.478		15.16
	Same; 3 in. from edge..................	Fine........	$\frac{1}{3.81}$	89.572	1.279	0.958	174.70	3.120	2.579	0.541		17.34
		Coarse. ..		90.096	0.829	0.971	175.90	3.180	2.999	0.181		5.69
		Average..		89.989	0.937	0.968	175.44	3.167	2.912	0.255		8.05
	Same; 4.5 in. from edge..................	Fine........	$\frac{1}{6.45}$	89.266	1.074	0.929	176.10	3.110	2.239	0.871		28.01
		Coarse.....		90.108	0.742	1.048	175.70	3.195	2.759	0.436		13.65
		Average..		89.984	0.787	1.032	175.75	3.184	2.689	0.495		15.55
	Same; shot, or granules............	....	....	....	....	....	....	3.000	3.039	Within the limits of error.		

NOTE.—The amount of copper taken in the analysis for antimony and arsenic was from 10 to 25 g., depending on the grade of the metal.

TABLE II.—*Alloys of Copper, Granulated, and Cast into Bars; Showing Various Nitric Acid Solubilities of Gold.*

Element Alloyed. 0.5 Per Cent.	Granules (Shot).				Bar-Drillings.			
	Gold.				Gold.			
	Assay-Method.		Difference.	Gold-Solubility.	Assay-Method.		Difference.	Gold-Solubility.
	Fire.	Comb.			Fire.	Comb.		
	Ounces Per Ton.	Ounces Per Ton.	Ounce Per Ton.	Per Cent.	Ounces Per Ton.	Ounces Per Ton.	Ounce Per Ton.	Per Cent.
Pb	1.934	1.8935	0.0355	1.84	1.954	1.6965	0.2575	13.18
Bi	1.927	1.890	0.037	1.92	1.933	1.789	0.144	7.45
Sb	1.900	1.832	0.068	3.58	1.890	1.676	0.214	11.32
As	1.894	1.826	0.068	3.06	1.874	1.592	0.282	15.05
Te	1.923	1.848	0.075	3.90	1.910	1.748	0.162	8.48
Se	1.935	1.856	0.079	4.08	1.935	1.860	0.075	3.88

TABLE III.—*Solubility in Nitric Acid of Gold in Miscellaneous Copper-Arsenic Alloys.*

Drillings From 3,000-g. Bars.				
Arsenic.	Assay-Method ; Gold.			Gold-Solubility.
	Fire.	Comb.	Difference.	
Per Cent.	Ounces Per Ton.	Ounces Per Ton.	Ounce Per Ton.	Per Cent.
1.680	21.381	21.245	0.136	0.636
1.110	2.018	1.965	0.053	2.626
0.767	11.081	10.827	0.254	0.290
0.511	7.9497	7.874	0.0957	1.204
0.369	6.214	6.140	0.074	1.191
0.225	3.112	3.014	0.098	3.149
0.117	1.576	1.524	0.052	3.300

TABLE IV.—*Copper (62 lb.) Melted in Crucible, Gold and Arsenic Added, and Left to Cool Therein in the Furnace.*

Position of Copper in Crucible.	Sample.	Ratio.	Arsenic.	Gold.			Gold-Solubility
				Assay-Method.		Differ-ence.	
				Fire.	Comb.		
			Per Cent.	Ounces Per Ton.	Ounces Per Ton.	Ounces Per Ton.	Per Cent.
Top third...	Fine.........	$\frac{1}{6.037}$	1.423	7.521	6.270	1.251	16.633
	Coarse		1.165	7.226	7.116	0.110	1.522
	Average.....		1.201	7.2679	6.9957	0.2722	3.745
Center third	Fine..........	$\frac{1}{6.835}$	1.013	6.522	5.955	0.567	8.694
	Coarse........		0.864	6.432	6.370	0.062	0.964
	Average. ...		0.883	6.4434	6.3170	0.1264	1.962
Bottomthird	Fine..........	$\frac{1}{5.574}$	0.959	6.422	6.040	0.382	5.948
	Coarse........		0.896	6.262	6.193	0.069	1.102
	Average.....		0.904	6.2863	6.1697	0.1166	1.855
Top, center, and bottom	Fines combined, melted and shotted...........	....	....	6.722	6.570	0.152	2.261

TABLE V.—*Electrolytic Copper (62 lb.), Melted in Crucible, Gold Added, and Left to Cool Therein and in Furnace.*

Position of Copper in Crucible.	Sample.	Ratio.	Gold.			Gold-Solubility.
			Assay-Method.		Differ-ence.	
			Fire.	Comb.		
			Ounces Per Ton.	Ounces Per Ton.	Ounce Per Ton.	Per Cent.
Top.................	Top.................	....	1.929	1.838	0.091	4.717
Center..............	Center.............	....	1.932	1.804	0.128	6.625
Bottom.............	Bottom............	....	1.879	1.756	0.123	6.546
Combined, and separated into fine and coarse	Fine................	$\frac{1}{5.460}$	1.852	1.314	0.508	27.760
	Coarse.............		1.894	1.834	0.060	3.168
	Average...........		1.887	1.758	0.129	6.836

TABLE VI.—*Lead* (99.15 *Per Cent.*) *Melted, Gold Added, Cast into* 9- *by* 9- *by* 9-*In. Cube and Left to Solidify and Cool in a Hot Brick Mold.*

Portion of Cube.	Gold.			Gold-Solubility.
	Assay-Method.		Difference.	
	Fire.	Comb.		
	Ounces Per Ton.	Ounces Per Ton.	Ounce Per Ton.	Per Cent.
Top	1.739	1.707	0.032	1.840
Center	1.777	1.749	0.028	1.576
Bottom	1.715	1.699	0.016	0.933

SUBJECT TO REVISION.

[TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS.]

The Bearing of the Theories of the Origin of Magnetic Iron-Ores on Their Possible Extent.

BY FRANK L. NASON, WEST HAVEN, CONN.

(New York Meeting, February, 1912.)

In the year 1904 an eminent Swedish geologist prepared a report on the iron-ore reserves of the world. His estimates follow:

Countries.	Tons.
United States,	1,100,000,000
Great Britain,	1,000,000,000
Germany,	2,200,000,000
Spain,	500,000,000
Russia and Finland,	1,500,000,000
France,	1,500,000,000
Sweden,	1,000,000,000
Austria-Hungary,	1,000,000,000
Other countries,	1,200,000,000
Total,	10,000,000,000

The iron-ore allotted to the United States by the Swedish geologist is barely sufficient to last 20 years at the present rate of consumption.

On the other hand (according to Edwin C. Eckel,[1] from whom the above and following statistics are quoted), there is hardly need for worry on the part of our iron industries for the next two centuries or more.

Upon the basis of careful work by the U. S. Geological Survey, in which Mr. Eckel has taken an important part, the following estimates are definitely, but by no means finally, given for the United States:

District.		Tons.
Lake Superior District,		1,500,000,000 to 2,000,000,000
Alabama,	red hematite,	1,000,000,000
	brown hematite,	725,000,000
Georgia,	red hematite,	200,000,000
	brown hematite,	125,000,000
Tennessee,	red hematite,	600,000,000
	brown hematite,	225,000,000
Virginia,	red hematite,	50,000,000
	brown hematite,	300,000,000
Total,		4,725,000,000 to 5,225,000,000

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[1] *Mineral Resources of the United States*, pp. 67 to 102 (1906).

These figures deal almost exclusively with red and brown hematites, which, as Mr. Eckel justly remarks, can be estimated with a possible error of from 5 to 10 per cent. Except in the Michigan and Wisconsin fields, magnetites do not enter at all into Mr. Eckel's estimate. Moreover, it is confined to known mines and districts, usually well exploited, and does not cover the possibilities of new fields in the States named, or of similar fields in other States. Mr. Eckel says:

> "If to this we add the ores occurring in the deeper levels in the States named, and also the red and brown ores of Maryland, West Virginia, and Kentucky, and the magnetic ores of the other Southern States, it is probably fair to assume that the total Southern ore reserve will amount to very nearly 10,000,000,000 tons, or five times that credited to the Lake Superior district."

On page 78 Mr. Eckel remarks:

> "The amount of Clinton ore in any area can probably be determined by a geologist within 5 or 10 per cent., and estimates of red ore reserves can, therefore, be made by competent men with a degree of accuracy impossible in dealing with magnetites, hematites, or brown ores of other districts."

To make a final quotation from Mr. Eckel's sane and comforting paper:

> "It is probable that careful exploratory work will develop magnetic iron-ores in New York, New Jersey, and Pennsylvania in quantities far in excess of anything usually considered possible in these States."

In my opinion Mr. Eckel could have included in his list not only most of the New England States, but the Appalachian States as well.

From the above citations, we may rest assured that an iron-ore famine does not immediately threaten the United States. The accuracy with which tonnage-reserves of the Clinton iron-ores can be estimated will hardly be disputed by any geologist. To show that deposits of magnetic iron-ores, whatever their origin, can be almost as accurately estimated, and that, with these ores added, Mr. Eckel's figures of possibly ten to twenty billion tons can probably be doubled, is the main object of this paper.

The theories as to the origin of magnetic iron-ores may be grouped as follows:

I. PRIMARY OR COTEMPORANEOUS (*i. e.*, sedimentary origin).

II. SECONDARY (practically vein-formation).

III. IGNEOUS (*i. e.*, segregation from molten magma).

Evidently, this classification is not perfect, in the sense that the three theories are mutually exclusive; but it will serve my purpose.

Given, or assuming, that the origin of magnetic iron-ores be ascribed to either of the above divisions, what bearing will the assumption have: (1) on the areal extent of any given occurrence; (2) on the ability of a competent geologist to estimate the probable volume of magnetic iron-ores?

In short, whether the magnetic iron-ores originated as sediments, as secondary, or as igneous deposits, are there any fixed relations between ore and country, or any characteristics of the ore-bodies, *per se*, which will enable a geologist to limit the extent of a given magnetic iron-ore field with the close accuracy possible with the Clinton ores?

This is the burden of the analysis which follows.

I. PRIMARY OR COTEMPORANEOUS ORIGIN OF MAGNETIC IRON-ORES.

This theory means that the magnetic iron-ores are a sedimentary rock-formation occurring in an uninterrupted chronological sequence. If it be accepted, what bearing does it have on the extent of the magnetic iron-ores? Does its careful study afford a means of predicting, within economic limits, the extent of these ores?

The sedimentary theory affirms that the rocks inclosing the ores were first a series of sediments: limestones, pure to highly impure; interstratified sands, fine to coarse-grained, calcareous or argillaceous; clays, calcareous and arenaceous, etc.

In this series may be interpolated one or more beds of nearly-pure iron-ore, either chemical or mechanical in origin, and one or more beds of impure to very impure iron-ore, of like origin. A railroad-cutting through earth layers, a section of a clay-bank and its overburden, or a rock-cutting or natural rock-section may show the sequence of deposition, and also how any given stratum thins and thickens, or changes along its strike from pure to impure, or even from slate to limestone or from limestone to sand or slate. Such a change from pure to impure ore, and *vice versa*, can be easily imagined, and is actually observed in Red Mountain, Birmingham, Ala.

Upon the deposition of a ferruginous series of rocks, may, and probably will, follow a series of rocks so free from iron as to be classed as non-ferruginous. We thus have foot- and hanging-wall rocks developed, as known to the miner. Between these walls will be the "rich" or "lean" ore, and rock which contains so little iron as not to be classed as ore at all.

Recurring to the actual formative period, it is important to note that the variation in the iron-content of the iron-ore series must be referred to the shore-line of the sea. The iron-series may be forming for miles along the shore. This will give one dimension of the iron-ore series, the length on strike; the second dimension, width, *i. e.*, distance sea-ward, will be determined by the period of time during which either subsidence has taken place, or the sea has eaten inland from the first shore-line.

To anticipate, somewhat, our conclusions: judging both by the iron-bearing gneisses and the magnetites themselves, the shore-line of the Archæan sea reached, on our Atlantic seaboard, from Ungava bay to the State of Alabama, a distance of over 2,400 miles. In New Jersey, these gneisses outcrop across their strike, a distance of 18 miles; and workable deposits of magnetic iron-ore occur not only on the extreme margins, but at many points between. On the sedimentary theory, the exposed rocks of the iron-bearing period exist inland at least 18 miles, with no diminution of iron except in variable amounts. Farther in New Jersey, to cite a single example out of many possible ones, there is a succession of superimposed strata, covering over 1,000 ft. of thickness at right-angles to bedding-planes. This instance can be supplemented and fortified by well-known occurrences of magnetic iron-ores, and their peculiar accompanying "gray gneisses," in SE. New York, in the Adirondack region, and in southern Canada from Port Arthur to the St. Lawrence river, in eastern Ontario.

On the assumption of sedimentary origin, we have, then, ocular proof in New Jersey that the geological period of the formation of magnetic iron-ores covered from West Point, on the Hudson river, to the Delaware, a shore-line 84 miles long, and that this shore-line advanced inward 18 miles.

Further, making no assumptions based on the entire thickness of the magnetite-bearing gneisses, we know, from actual superimposed beds of magnetic iron-ore, that one ore-series is more than 1,000 ft. thick; that in this series are five workable deposits of ore; and that these deposits average more than 10 ft. in thickness from foot- to hanging-wall.

From the "primary," "cotemporaneous," or "sedimentary" origin of magnetic iron-ores, as here stated, it follows as a corollary that the iron-ores are conformable, that is, more or less parallel to their inclosing rocks, and that probably the iron-ore stratum will be more or less co-extensive with the inclosing rocks, after the manner of coal-seams or beds.

Variation in Sedimentation.

Sedimentation, or cotemporaneous deposition of iron-bearing strata, may be effected by mechanical or chemical processes, or both together. One part of a given stratum may be wholly clastic in origin; another part may be wholly chemical; or any given part, or the entire stratum, may combine the two factors in varying proportions. Which of the two has been the dominant one, has an important bearing on the possibility of estimating the extent of the magnetic iron-ores.

1. *Mechanical, or Clastic, Origin.*—This requires three conditions: (1) an open sea-front; (2) granitic, gneissic, or eruptive rocky shores, either disintegrated or solid or both; and (3) iron-ore in the rocky shore-line—and this iron-ore must be in solid grains or masses, as magnetite, hematite, or limonite.

An open sea is an indispensable assumption: otherwise there would be no waves to "pan" the lighter sand, etc., leaving the heavier iron-ore in windrows on the beach. A shore of granites, gneisses, or eruptives must be assumed, since limestones, clay-slates, sandstones, etc., rarely carry grains of iron-ore in appreciable quantities.

The iron must exist in these rocks in the form of magnetite, hematite, or limonite, in grains or masses, otherwise there could be no mechanical concentration or panning. The extent of the deposit of clastic iron-ore would depend directly on the percentage of iron in grains and masses in the shore-line.

It is a well-known fact that grains of magnetite and hematite are not easily dissolved by natural agencies. The occurrence of magnetite in sea- and lake-sands; in mountain streams and on inland plains; the occurrence of grains and crystals of magnetite, still highly magnetic, in the rotted gneisses and granites of the non-glaciated regions, etc.—give sufficient evidence in this line, and lend great plausibility to the theory of the clastic origin of magnetites.

2. *Chemico-Sedimentary Origin.*—For this process, we must assume conditions the very reverse of those assumed for the clastic phase. Instead of an open sea, it requires closed lagoons, low, generally or periodically flooded marshes or great inland lakes or seas; since chalybeate waters, however heavily burdened, pouring into an open sea would be so widely dissipated that no economic ore-body would be formed.

Iron is one of the most widely and abundantly distributed of the elements. According to Dr. Clarke, metallic iron constitutes 4.49 per cent. of the earth's crust. Therefore, under normal conditions in waveless, inclosed or sheltered waters, soluble iron would be leached to the extent of 4.49 per cent. of the rock-mass, of whatever nature, and precipitated more or less evenly and regularly over the entire bottom of the lagoon. While the "average per cent." of Dr. Clarke includes the entire rock-crust of the earth, it is evident that a large part of that crust is iron-free. This rather militates in favor of our assumption. Since the basis of our discussion is an actual ore-deposit, if the deposit be ascribed to chemical solution and precipitation, the decomposing rocks feeding the deposit must have had at least the 4.49 per cent. average of Dr. Clarke.

The granites and gneisses would contribute only their soluble iron in biotites, hornblendes, etc. The shores being tideless and waveless, magnetite, hematite, garnet, and silica would remain *in situ;* iron, lime, magnesia, and alkalies would be leached out and carried into lagoons, there to be precipitated or in part to remain in solution. It is, however, possible to imagine that the readily-soluble minerals of gneiss, granite, or eruptives, and, to a greater or less extent, alumina, might be wholly eliminated. There would thus be left the insoluble magnetite and hematite, in grains or even in large masses, to form "residual" beds; and these would be supplementary to the deposits strictly chemical in origin. It is hardly necessary to state in detail how chemical and mechanical force could each play a part in the same stratum.

Economic Bearing of the Sedimentary Theory.

The Clinton iron-ores are regarded as cotemporaneous sediments, and likewise as chemical precipitates. Experience has taught that these ores are nearly coextensive with the containing rocks. For these reasons, in a given locality, a competent observer can estimate the tonnage within an error of from 5 to 10 per cent. If, therefore, the chemico-sedimentary origin of magnetic iron-ores can be proved, or even shown to be reasonable, there is no reason why a careful study of magnetic iron-ore fields should not permit an equally accurate estimate.

Without going into detail, we must confess that, if the magnetites are of wholly clastic origin, the above margin of from 5 to 10 per cent. is too small to be safe. For a combination of the two origins, chemical and clastic, the chances for accuracy in a given district are about equal with those attending the Clinton ores. In both cases, the fundamental principle is simple.

The magnetic iron-ores are a rock-stratum; this stratum outcrops in rocks of its own age, and these rocks cover thousands of square miles; there is thus no known or imaginable reason why one should limit the extent of a magnetic iron-ore to a few feet in length and a few feet in depth.

Statistics.—Whether or not the true reason for the common skepticism as to our magnetic iron-ore reserves is indicated by the following figures, they are suggestive:

As late as 1880, the Appalachian region produced 5,667,348 tons of iron-ore, or 67 per cent. of the total of 8,487,250 tons. Of this total, 30.02 per cent. was magnetite. In 1910, the total iron-ore production of the United States was about 53,000,000 tons, of which about 5 per cent. was magnetite.

In 1880, magnetic iron-ores were chiefly mined in New York and New Jersey. The ore-shoots then economically workable were small, because all ores carrying less than 50 per cent. of iron were unmarketable. The total product of each mine was small, and the cost high; consequently, mining to great depths was economically impracticable; and a "pinch" usually caused a shut-down. Exploration was limited to rich shoots and moderate depths.

Four years later the great high-grade deposits of Minnesota began their shipments.

These facts may explain why outcrops of magnetic iron-ore are supposed, generally, to cover no more than 6,000 sq. ft., *i. e.*, 600 by 10 ft., and to become exhausted at less than 1,000 ft. in depth on an average. Drop the required iron-content of an ore from 55 to 35 per cent.; and, to my personal knowledge, there are magnetic iron-ore outcrops from 1 to 7 miles long. In actual thickness of the bed, these bodies vary from 2 to over 100 ft. In extension on the dip, they have been proved less than 1,000 ft. on an average, but in one instance over 3,000 ft., and the boundary not there demonstrated. By the sedimentary theory, then, the magnetic ores, as well as the Clinton, should be given an inclined depth proportionate to the length of outcrops.

II. SECONDARY ORIGIN OF MAGNETIC IRON-ORES.

On this hypothesis (whatever the nature of the inclosing rocks, igneous eruptives, metamorphosed sediments, or metamorphosed sediments into which sheets of molten eruptives have been thrust or over whose surfaces molten eruptives have been poured, or both), the magnetic iron-ores were deposited after the rocks were formed, and can only have been so deposited by chemical action, that is, the solution and deposition of the iron, by and from circulating waters.

Such waters may have been derived from either or both of two sources. The first is known as "magmatic" water in eruptives, or as "rock-water" and "water of crystallization" in the case of sedimentary rocks. The second is "meteoric" water, entering from the earth's surface into the underground circulation.

As to the source of the iron to be carried by these waters, it has already been pointed out that metallic iron constitutes nearly 4.5 per cent. of the rocks of the earth's crust. A mass of rock 1 mile square and 500 ft. thick weighs about 1,072,200,000 tons. At 4.5 per cent. of metallic iron in such a block, the content of metallic iron will be 48,249,000 tons, equal to about 83,700,000 tons of magnetite. This would furnish material for a pure, solid magnetite ore-bed 1 mile square, and more than 20 ft. thick.

As we are dealing with the same rock-formation as before, but assuming eruptives or a combination of eruptives and sediments, instead of sedimentary origin, the areas in both cases will be coextensive.

Water and iron having been accounted for, the next step is to find room for deposition.

Two cases will arise here, as has already been suggested. The rocks are either sedimentaries, penetrated or overflowed by eruptives, or they are wholly igneous or eruptive.

In the first case, we may suppose that the original sediments were alternating beds of sandstones, shales, and limestones. Limestones are not only readily soluble, but as the limestone is dissolved in iron-bearing waters and carried away, iron is deposited in its place by a simple chemical reaction. The purity of the deposited iron-ore would depend on the purity of the limestone. A pure limestone would be wholly dissolved and replaced by pure iron-ore, while from an impure limestone all silica, alumina, etc., would be left behind, to mingle with the ore.

We are familiar with the fact that a limestone bed is rarely uniform in composition. Lenses or bunches of impure limestone occur with more or less regularity. Hence, if 1 square mile of rock were originally limestone, a vertical section of it, after the lime had been dissolved out and replaced with iron-ore, would show lenses of pure ore passing into strings of pure ore interbanded with lean—just the conditions, in fact, which the magnetic iron-ore miner finds in his mines to-day.

In the second case, where the entire rock-mass is a succession of eruptives, the rock, instead of easily-soluble limestone, would consist of less easily decomposed feldspars, micas, hornblendes, pyroxenes, etc. The end, however, would be practically the same as with the limestones.

The decomposition by water of solid crystalline rocks is abundantly illustrated by the gneissic and granitic rocks of the unglaciated Southern States, where sections of rotted rocks many feet thick are exposed. As to the power of water to dissolve the iron from these rocks, and the later precipitation of this iron, the 725,000,000 tons of minable "brown ore" which Mr. Eckel estimates in Alabama, Georgia, Tennessee, and Virginia alone, is conclusive proof.

Reviewing, briefly, the salient points in the secondary origin of magnetic iron-ores, we find that, whether the rocks containing the ores originated as sediments or eruptives, they cover many thousands of square miles. As is demonstrated in the case of the Clinton ores, the extent of the ores is not limited to the outcrops, but conditions similar to the outcrops may reach at right angles to the outcrop for thousands of feet, limited only by the extent of circulating water. We find an abundant water-supply in magmatic, rock, or metoric water. We find channels through which this water can readily circulate. We find an abundance of rocks and minerals which are dissolved or decomposed by water. We find in the rocks themselves an abundant source of iron and in a readily-soluble form.

Therefore we can but conclude that, as in the case of sedimentary origin, so in the case of eruptives there is no visible reason why the magnetic iron-ores should not be coextensive with the special rock-formation, as is the case with the Clinton iron-ores.

III. Igneous Origin of Magnetic Iron-Ores.

This theory appeals to the well-known fact that crystals and crystalline grains of magnetite and hematite are constituents of nearly every crystalline eruptive. The percentage of metallic iron will exceed 4.5 per cent. We have already seen that this means an equivalent of 83,700,000 tons of magnetite in a square mile of rock 500 ft. thick. The problem is, to account for the assembling, not of all, but of a small part of this magnetite in the form of workable ore-bodies.

The principle involved is very simple and can be readily observed by every smelter of iron- or copper-ores. Fragments of matter held in mechanical suspension in a fluid or semi-fluid of lower specific gravity will settle and gather, so long as the suspending medium remains sufficiently fluid.

In copper-smelting, slag and matte, intimately mixed, are drawn from the furnace into pots. So long as the slag remains fluid the matte settles and collects in the bottom of the pot. When the slag becomes viscous in cooling, suspended grains of matte will be fixed in position, and unable to enter the main mass of matte. To apply this principle to the segregation of magnetic iron-ores by the igneous theory we have only to assume that all igneous rocks were at one time more or less fluid, and that they retained their fluidity for a long period. Moreover, we know that magnetite and hematite occur in all eruptives, and that both are nearly twice as heavy as the rock-mass in which they occur.

Therefore, according to this theory, the igneous rocks containing the magnetic iron-ores are but a gigantic slag-pot, from which the matte, in this case iron-ore, has, or may have, settled. One might jump to the conclusion, then, that every proved eruptive rock would contain a body of iron-ore. But the reasons why this sweeping conclusion is not at all warranted are natural and self-evident.

In the first place, as in the case of the furnace, all eruptives are not equally fluid. It might therefore happen that the weight of the iron grains would not be sufficient to cause them to settle through the viscous fluid. This alone would suffice to forbid the general expectation of an iron-ore body in every eruptive. Unequal fluidity of the mass as a whole might give rise, in one locality, to strings or bunches of ore wholly wanting in another.

Without pursuing further this obvious line of thought, we must consider the shape of individual masses of ore. Gathering in the manner that has been pointed out, the shape of the mass of ore could be nothing else than very irregular. Pure ore would be succeeded by pure rock and this in turn by lean ore and again rock. Nor, further, have we any clue as to how far in length, breadth or depth conditions favorable to segregation would obtain. With this cause dominant in the origin of iron-ores, obviously no safe estimate of the volume of reserve magnetites can be made. However great the outcrop, the extent of the ores depends upon the operation of conditions the extent of which there is absolutely no known method of determining. This much, however, can be predicted with reasonable certainty: the igneous iron-ores occur in undoubted igneous rocks which can be recognized almost unerringly. Moreover, the ores occurring in these rocks are, almost without exception, high in titanium.

All iron-ores high in titanium are exclusively in an eruptive rock generally called gabbro. Without exception, these ore-bodies are very irregular in outline. Every ore-body begins and ends abruptly. There is no leader, stratification, schistosity, bedding, or strings or lenses that will lead from one deposit to another.

There are many features peculiar to the titaniferous iron-ore deposits. Whatever the origin of the non-titaniferous iron-ores and their inclosing rocks, the gabbros are found overflowing or overlying them, intruding in the form of sheets or cutting them in the form of bosses or knobs. The gabbro rocks and their titaniferous ores are younger than the non-titaniferous rocks.

Moreover, in the Adirondack region of New York, where iron-ores high in titanium occur in close juxtaposition with others practically free from this element, the titaniferous ores lie exclusively in the igneous gabbro, while the ores lying in the so-called gneisses are wholly unaffected by titanium. The most natural inference is that the two ores are wholly unrelated in origin, as to time, though both may be eruptive. Yet it is not certain that they may not be cotemporaneous. But, even in that case, they are sharply distinguished. The segregated titaniferous ores occur in an eruptive gabbro; the non-titaniferous ores in a distinct gneiss; the former as isolated pockets, sometimes of great size; the latter as extensive sheets, so regular and conformable to the schistosity of the inclosing rocks as to make prospecting, areally, for an extension of a given outcrop a safe business. Finally, the titaniferous iron-ores contain (in addition to titanium) chromium, vanadium, and nickel, from which the non-titaniferous ores are relatively free.

Hence, in a district where gabbro rocks are dominant, however extensive the showing of iron-ore in these rocks, no estimate of ore beyond three visible dimensions can be made; while in a district in which the dominant rock is the gray gneiss of known iron-mines, the possibilities of magnetic iron-ore reserves are bounded only by the limits of the gneisses. Of course this rule can be applied only in accordance with sound judgment based upon adequate experience. The titaniferous ores are undoubtedly of eruptive origin. Whether the non-titaniferous ores are primary or secondary, the final limits of a given horizon or stratum have never yet been found.

In conclusion, there is one possible origin of the magnetic iron-ores which has not been generally considered.

In the province of Oriente, Cuba, there are deposits of iron-ore that literally cover square miles of territory, save for division by drainage-streams. One belt is, to my personal knowledge, more than 10 miles long by 6 miles wide. The deposit occupies the surface and extends from the "grass roots" down to a depth varying from a few inches to more than 100 ft. It is supposed to have originated by a process the very reverse of commonly accepted methods. Here a rock, originally high in iron, has been leached of its refractory silica, leaving as a residue, alumina, and carrying from 20 to 50 per cent. of metallic iron. This ore is not confined to Oriente, but is found extensively in Camaguey, covering large areas, and more than 100 ft. thick.

This variety of iron-deposit is supposed to have been caused by the chemical activity of meteoric waters in a warm, moist climate.

But our world was, during one long period of its history, probably subject, over its whole area, to precisely these conditions. There is thus a possibility, not to say a probability, that many iron-ore deposits may have been formed in this way. All that Cuba would need, to convert into magnetite her deposits of soft ores, would be a mantle of soil and a later metamorphism; and it is not difficult to imagine similar conditions as having resulted in our extensive iron-bearing gneisses with their deposits of magnetic iron-ores.

Viewed from any stand-point of genetic theory, then, we can only come to the conclusion that our magnetic iron-ore reserves are very great, and that, in a given district, their extent can be almost, if not quite, as accurately estimated as in the case of the Clinton ores.

When we consider, further, the fact that along the Atlantic coast, for 2,400 miles, from Ungava Bay to Alabama, and for an average width exceeding 100 miles, not only are the iron-bearing gneisses known to occur, but in this area many outcrops actually do occur from 1 to 7 miles along the strike, there is as yet no known reason why a proportionate depth cannot be safely assumed.

CONCLUSION.

It is hardly necessary to say that the foregoing argument does not deal with the question of ore-reserves "in sight," or actually indicated by recognized outcrops only. It does not warrant sanguine estimates of the value of particular undeveloped properties. It simply seeks to show that we are justified in assuming for our magnetic iron-ores a potential extent, in something like the way we calculate upon other iron-ores; and that, in this respect, the estimates of some geologists and the real or affected fears of others are alike groundless.

[TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS.]

Geology of Harrison Gulch, in Shasta County, California.

BY H. E. KRAMM, ITHACA, N. Y.

(New York Meeting, February, 1912.)

During the summer of 1911, I had the opportunity to study in detail the geological conditions of what is known in northern California as "Harrison gulch," in Shasta county. Though the district, as a whole, that is, the western part of Shasta and the eastern part of Trinity county, has contributed during the last 15 years a considerable share of the gold-production credited to Shasta county, very little is known of its geology, perhaps because the country is covered with a dense growth of brush which makes exploration and prospecting extremely difficult.

Topographically the district presents a succession of ridges trending approximately N–S., and alternating with deep narrow canyons. The drainage is partly into the Sacramento, partly into the Trinity river.

Climatic conditions in summer time are almost ideal. It is true, the thermometer may climb to 115°, but the air is dry and the heat is bearable, especially as the nights are always cool. It seldom rains during the summer months, and creeks and springs dry out towards the end of the summer. The winter is severe, lasting from November to April, and usually a thick layer of snow covers the mountains.

By reason of the character of its gold-deposits, the region is known to miners as "the pocket country." It contains a series of eruptive rocks intruded into slates, cherts, limestones, and conglomerates. Dikes, laccoliths, and batholiths are frequent. In a day's ride it is possible to find all the common igneous rocks from the very acid to the very basic. Serpentines are especially abundant. The sedimentary rocks are, as rule, broken and shattered as well as folded, and show the effects of intense metamorphism. "Pocket-hunters" speak of the contact of diabase and slate as the "pocket-contact." Some rich gold-pockets, however, have also been found in the diabase.

Besides the gold, there are indications of quicksilver and copper. Cinnabar is often present in the concentrates of the placer-gravels, and can invariably be traced to a contact of serpentine and shale. It seems, therefore, that the conditions of its occurrence are similar to those of the central coast ranges of California, where, with a few exceptions, it is found in decomposed serpentine near the contact of that rock with shale or sandstone.

Up to the present time quicksilver has not been found in paying quantities in Shasta county, but such a discovery would not be surprising.

Indications of copper are stains of malachite and chalcopyrite. They are, in all probability, due to the leaching of copper-minerals which most basic eruptive rocks carry, and which are redeposited. They are especially abundant in quartz veins found in slates and basic eruptive rocks; and since the quartz may also be a carrier of gold, the presence of the stains may make the cyanidation of such ore a difficult problem.

The area described in detail here comprises about 6 sq. miles, including that part of Harrison gulch in which the property of the Goldhill Consolidated Mines is located, and also the town of Knob.

Knob is reached from Redding, which lies to the northeast, by daily stage which leaves at 6 a. m., passes through Ono and Igo, and in summer time manages to reach its destination in the evening. A road also makes connection with Red Bluff. Trails lead to Weaverville in the north and to other points. Redding and, to some extent, Red Bluff are bases of supplies.

Figs. 1 and 2 give an idea of the topography and geology. Harrison Gulch creek, which empties into Cottonwood creek, a tributary of the Sacramento river, has its headwaters within the area. Its course is approximately N–S., and it has cut a steep canyon, the walls of which rise in places more than 1,000 ft. above the creek-bed. Numerous smaller streams have left their imprint upon the topography, the rugged appearance of which is heightened by the dense growth of brush and an abundance of pine and fir timber.

The following rocks appear in this district:

1. A shale of siliceous character, reddish yellow in color, fairly hard, and showing a pronounced cleavage. Near its con-

tact with a diabase, which is intrusive, it becomes slate, its color changes to a greenish brown, and it carries considerable amounts of free carbon. Interbedded with it are a more or less iron-stained chert and lenticular bodies of crystalline limestone of great purity. The former appears to represent an intense degree of metamorphism of the shale.

2. A conglomerate, made up of pebbles of chert which vary in size from a diameter of several inches to that of a grain of sand. It is well cemented by a siliceous material. A friable shale, interbedded with the conglomerate, contains, near Knob Hill, north of Knob, a fossil flora.

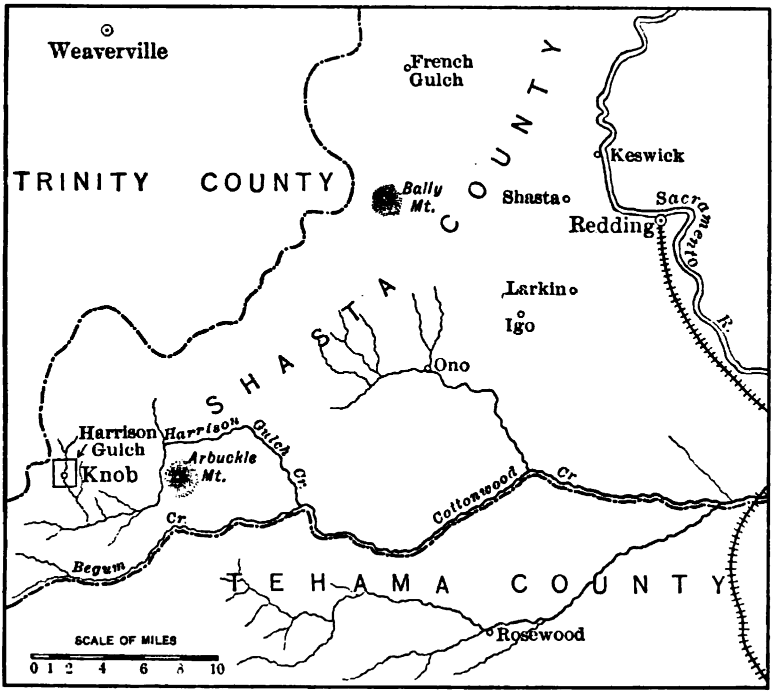

FIG. 1.—SKETCH-MAP OF SHASTA COUNTY, CAL.

3. Serpentine derived from a peridotite which is intrusive into the shale first named above. It ranges in color from dark green to brown, according to its state of decomposition. As a general rule, it is fairly well preserved, and shows several of the original minerals of the peridotite still intact.

4. A diabase which seems to grade on one hand into a gab-

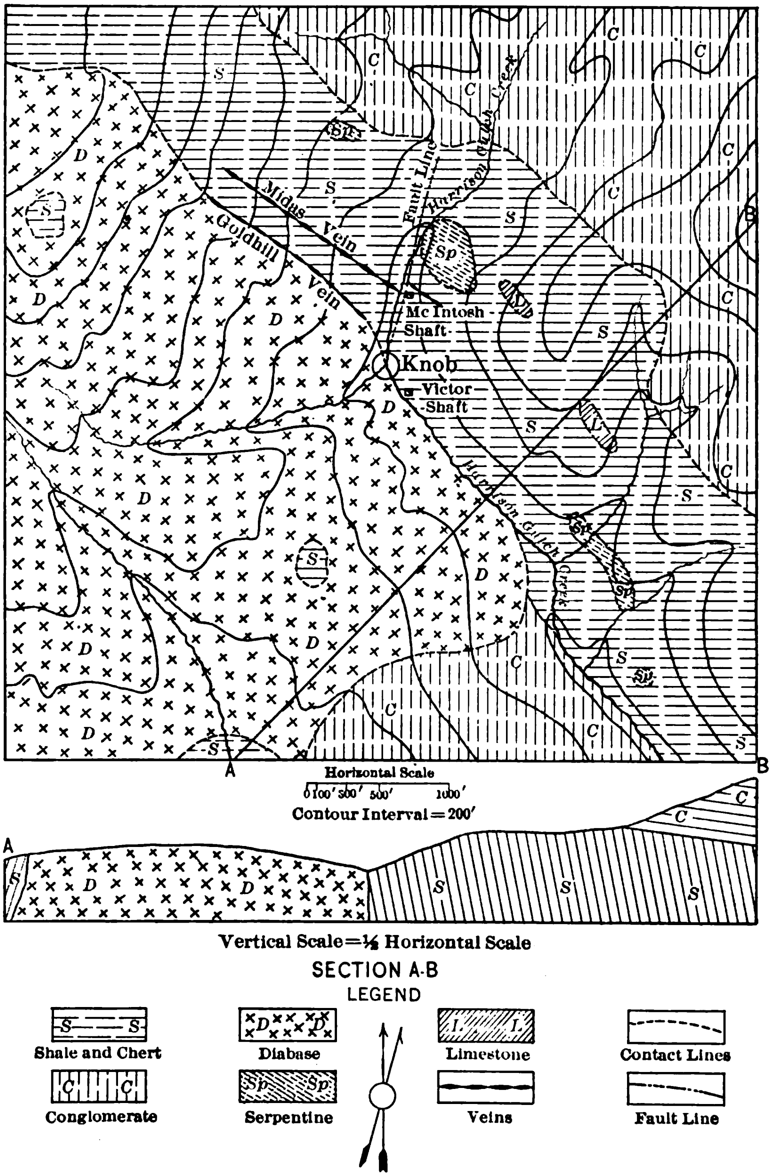

Fig. 2.—Geology and Topography of Harrison Gulch, Shasta County, Cal.

bro, and on the other into an andesite. It is somewhat decomposed and intrusive into the shale and conglomerate.

The shale, limestone, and chert were deposited first. Their angle of dip is high—approximately 70°. An unconformity exists with the overlying conglomerate.

The area investigated gives no clue as to the geological age of the rocks. They seem, however, to correspond in every respect with the Clear Creek volcanic series and the Bragdon formation, consisting of slates, quartzites, and conglomerates.

Geology of the Deposits.

The intrusion of the eruptives has given rise to conditions favorable to the formation of quartz fissure-veins. Stringers can be found traversing the slate and the diabase at many places. The two important veins mined for gold are known as the Midas and the Gold Hill.

For sake of description we may divide all the quartz veins of the area into three types.

The first type is represented by the Midas vein, a fissure in the slate, which has been worked to a depth of about 1,500 ft. It strikes approximately NW–SE., and dips about 80° S. Its width is variable, with a maximum of 3 ft. The walls are composed of broken and shattered minute angular fragments of slate, recemented by chalcedony. This condition is not confined to the wall-surface alone, but extends a considerable distance into the rock. It is scarcely visible to the naked eye, but can plainly be seen under the microscope.

The ore-minerals consist of native gold (alloyed with some silver), iron pyrites, and tetrahedrite. A thin gouge is sometimes present; more often the quartz rests directly on the slate wall.

The ore-body has undergone extensive faulting. Besides numerous faults of minor importance, among which there seems to be no system, there are two of considerable magnitude.

The first is known as the "header and gutter"—a horizontal displacement, averaging about 5 ft., which can be traced from level to level without serious difficulty, and which is important because it has been instrumental in the concentration of the gold. Below it, the values decrease rapidly; above it, they are high.

The second fault is more extensive. It is a normal fault striking approximately N–S.—that is, nearly at right angles to the strike of the vein. Its course is about that of Harrison Gulch creek near the McIntosh shaft. It dips 80° E. This fault caused a downward movement of 200 ft. of the country east of the creek, with a horizontal displacement to the north of about 200 ft. It is easily recognized and traced. Its walls at places stand 6 ft. apart, the space being filled with fault-breccia, gouge, and broken fragments of the ore-body. At other places the walls touch each other.

The "header and gutter" has been found to continue in that portion of the vein which was displaced by the second fault.

The second type of veins is represented by the Gold Hill, which differs from the Midas in that it is located along the shear-zone caused by the intrusion of diabase into the slate. It is irregular in dip and strike and eccentric in metal-content. A gouge of talc is often, but not always, present. Besides the ore-minerals found in the Midas vein, chalcopyrite in small quantities and native copper are present, the latter, however, being exceedingly rare.

Where no vein exists between the two rocks, the contact is sometimes sharply defined. On one hand is slate, on the other diabase, with a definite boundary-line. Sometimes, however, it is not well defined, but one rock so gradually passes into the other that even the microscope does not give satisfactory information as to the mineral content, the rock being much decomposed and showing thermal effects.

The third type embraces veins entirely in diabase. Their origin is probably due to the cooling of the intrusive rock and consequent shrinkage, which caused fissuring. They are very irregular in occurrence and shape. It is of interest that up to the present these veins have proven barren with the exception of those which lead into the shear-zone and contain values near this zone.

The distribution of ore-minerals in the three types of veins makes it certain that the slate was an influential factor in their deposition. Mineral-bearing solutions, meeting the carbonaceous waters issuing from the slate, were forced to deposit their mineral content. This explains why the third type of vein is mineralized near the shear-zone, but barren some distance away from it. It may also explain why the ore-minerals are most evenly distributed in veins of the first type, where both walls are slate and could act as precipitating-agents.

That the carbon acts as a powerful reducing-agent is demonstrated by the occurrence of native copper along the shear-zone. This carbon has proved a hindrance in the cyanidation of the ore.

In mining the ore, it is not possible to avoid including a certain percentage of slate. The ore is crushed and amalgamated, and the tailings are cyanided. While the percentage of copper present is not enough to interfere with the cyanidation, the free carbon entering with the slate has caused serious difficulty. It was at first believed that the sulphides contained in the ore were the objectionable elements, and roasting was tried with success. As a matter of fact, however, the roasting transformed most of the carbon into carbon dioxide and thus freed the ore of the objectionable reducing-agent.

In closing, I wish to acknowledge my obligation to H. L. Waste, Manager of the Goldhill Consolidated Mines Co., with whose permission this paper has been prepared.

SUBJECT TO REVISION.

[TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS.]

Abrasion and Dust-Losses in Ore-Drying.

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(New York Meeting, February, 1912.)

THE problem of drying ores is one that most mill-engineers are sooner or later called upon to meet, and it may be timely to point out some of the difficulties resulting from such operations from purely a metallurgical and economic stand-point. We mean, in the following pages, to discuss primarily the effect of such operations on the ore itself, without particular reference to the mechanical or cost features of the subject. That an appreciable tonnage of the ore fed to the usual form of drier, either rotary or stack, is lost, is fairly well appreciated, but we believe that only a few mill-managers know just what their drier-losses amount to, and we have found that in most cases they console themselves with the fact that the drier-smoke is not very much discolored, and it is generally believed that the draft is not sufficient to carry away more than a negligible percentage of the values.

The drying of ores is becoming an important factor since the more general application of magnetic, static, and dry-table treatment. Its importance in the field of metallurgical operations warrants a close examination into its effects. This subject has been repeatedly brought home to us, and while the drier-losses, in every case examined, have considerably exceeded even the most generous estimate, an attempt at a recovery of these values is rarely made. We have found that the losses resulting from drying-operations can be effectively recovered, at a low cost, and believe that the publication of the results of a number of investigations along these lines will be of interest and benefit to the profession at large.

To dry an ore quickly and effectively, it must be brought into actual contact with hot gases or air. The rapid drying is effected by the fact that the hotter the gas the hungrier it is for moisture. A consideration of the following data indicates the rapidly increasing moisture-carrying capacity of air or gas, at atmospheric pressure, as the temperature increases:

Temperature. Degrees F.	Weight of Water-Vapor Mixed with 1 lb of Air. Pounds.
32,	0.003744
82,	0.023526
132,	0.118548
182,	0.728012
192,	1.253190
202,	2.855070
212,	Infinite.

The types of driers usually employed are: (1) stacks of brick or iron in which the ore falls over baffle-plates, against the ascending current of hot gases, and (2) horizontal rotary tubes, in which the ore is propelled, due to the increasing diameter of the conical shell, or to a slight pitch when the tube is a cylinder; or to either of these in addition to a series of spiral angle-plates attached to the interior of the shell acting as a screw-conveyor.

In the rotary driers, it is usual to have lifting-angles on the inner wall of the shell for picking up the ore and carrying it to a varying height, allowing the material to fall to the bottom of the cylinder through the current of hot gases. Thus a myriad of surfaces is presented from which the moisture may be evaporated. In the stack-drier, the same effect is produced when the ore drops from baffle to baffle, thereby being spread out into a rather thin layer.

While under these conditions the drying-factor is a fairly efficient one, yet two other effects, not desirable, result. The first of these is abrasion, while the second is the loss due to the draft carrying along fine particles of value. The abrasion-factor is practically the same in the two types of driers unless in the rotary type the passage of the ore is very slow and each piece is permitted to fall many more times than it would in passing from baffle to baffle in the stack drier. That the abrasion in the drying of sulphide ores is a large factor, and that as a rule the mineral values abrade the more readily, is generally admitted; but few engineers realize the extent to which driers act as a gravity impact-crusher or tumbling-mill. If the feed to a rotary drier is carefully examined and compared to the discharge, the shape of the pieces of ore will be found to be quite different; the feed being sharp-edged and rough, as against the discharge showing many smooth surfaces, while at the same time the rough and sharp edges have almost wholly disappeared. Abrasion is the cause, and of course the heavier the rock the greater will be the impact, and consequently the greater the abrasion. There is, therefore, an economic limit to the size of the material to be dried, entirely aside from the factors of damage to plates or shelves, or the danger of not thoroughly drying through to the core, in the case of porous material. On the other hand, a fine feed has the decided disadvantage that the operation produces much dust, resulting in a great loss.

Such drier-losses first made themselves uncomfortably noticeable to us in the operation of a dry milling-plant in Utah handling lead-silver ore in which two-thirds of the lead was in carbonate form, while fully 10 per cent. of the silver-values occurred as halogen compounds.

The ore was crushed to about 1.5 in. in size before entering the drier. This consisted principally of a conical shell 24 ft. long and about 4.5 ft. in diameter, rotating at 8 rev. per min. The draft was just sufficient to take care of a light fire, all that was necessary, since the ore carried but from 5 to 8 per cent. of moisture. Even under such conditions and after taking out the alternate lifting-angles, the gases issuing from the top of the stack showed, by their color, that an appreciable quantity of fine dust was being lost. The back connection of the drier consisted of a large chamber, the flue from which to the stack was very liberal in cross-section. Hence the flow of the gases was so very slow that most of the dust was permitted to settle in the chamber and in the bottom of the smoke-stack. A fairly close approximation of the distribution of the ore after it was dried is found in the following:

Products.	Weight. Per Cent.	Assay.		Percentage of Total.	
		Pb.	Ag.	Pb.	Ag.
Total feed, ...	100	10.35	9.68	100	100
Dry discharge, ...	96.2	9.77	9.34	90.7	95.40
Settled in chamber, . . .	1.77	35.30	10.30	6.04	1.94
Settled in smoke-stack, . .	0.76	18.50	8.60	1.36	0.69
Lost in gases, ...	1.27	14.30	15.20	1.75	2.04

From the above it will be seen that of the total feed to the drier, 3.8 per cent. by weight was withdrawn by the draft, corresponding to a loss of 9.3 per cent. of the lead-content and 4.6 per cent. of the silver-content. At the then prevailing metal prices, 7.6 per cent. of the total value in the ore was taken out by the draft. Of this, the greater portion, however, due to the generous dimensions of the settling-chamber, flue, and smoke-stack, was recovered in the form of a somewhat finely-divided product of sufficiently high grade to be mixed with the concentrates, resulting in a final loss of unrecovered material, going off as an impalpable powder in the stack-gases, of 1.27 per cent. of the total lead and 2.04 per cent. of the total silver, or a money-value of 1.85 per cent. of the total values in the ore.

Had it not been for the precautions employed in operating the drier and crushing no finer than 1.5 in., the final losses would have been much heavier and would have necessitated treating the drier-gases for a recovery of the dust. The slow movement accounts for the reduction in lead-tenor as the products grow finer, while the silver-values manifest their finely-divided form by their presence to the greatest value in the dust of the discharge-gases.

In the case of a cyanide-plant in southern Utah, crushing gold-ores dry, prior to cyanidation, the ore came from the mine in a very wet state. Consequently four rotary driers were driven hard to effect drying the tonnage necessary. The ore was crushed to but 2 in. in size before going to the drier-plant. The smoke issuing from the stack was light gray in color, due to the quartz, and on quiet days settled its burden over the immediately surrounding country. From 5 to 6 per cent. of the tonnage was lost regularly in this operation; and assays of the dust contained in the gases as issuing into the atmosphere ran 0.48 oz. of gold and 3.48 oz. of silver, derived from an ore carrying 0.31 oz. of gold and 0.15 oz. of silver. This dust was all finer than 150-mesh, but could readily have been recovered and treated. The loss suffered by this drying-method amounted to approximately 7.81 per cent. of the values in the ore.

Two other notable cases may be mentioned in passing; one, in northern Norway, where mixed magnetite and hematite ore carrying 35 per cent. of iron was being milled on a very large scale. The rock, after having been crushed to about 3-in. size, passed through a set of stack driers, which at certain times in the year, when snow and ice came down with the ore from the quarries, it was necessary to operate with forced draft. While the dust-loss was not actually measured, it is safe to say that it was not less than from 7 to 10 per cent. by weight. In addition to this, the assay-value of the dust was considerably higher than that of the crude ore, since the specular hematite, being friable, suffered the greatest abrasion. The surrounding country for hundreds of feet from the drier-house was covered with fine iron-ore, and the smoke from the drier-stacks was a dark brown and just as dense as that issuing from a chimney directly after boilers have been fired with soft coal. The trouble in this case was a lack of drying-equipment; and since the time that ore is exposed to the action of the gases in a stack drier is not readily controllable, a sharp increase in moisture-content of the ore necessitates driving the driers with forced draft, resulting in a very much heavier dust-loss than would be suffered if the upward-moving gas-current had a lower velocity.

Another instance in one of the far western States, where sulphide losses were heavy, may be mentioned. The material treated was a middlings product consisting of a zinc-blende and iron-pyrite, practically all of it finer than 16-mesh, and carrying from 20 to 30 per cent. of moisture. The separating process used, being a dry method, required the drying of the feed so that it would lend itself freely to fine sizing. The type of drier used was a rotary cylinder through which the gases passed in contact with the ore, and fearing a considerable drier-loss, a settling-chamber was installed between the back connection of the drier and the smoke-stack. The deposition of fine sulphide-dust in the settling-chamber was relatively very large, but frequently this was not deposited in the dry state, since a reduction in temperature of the gases in this chamber permitted some of the moisture to condense, and left a large portion of the settled product in the form of a thick sludge. Much of the fine dust escaped with the gases, as was evidenced by the color of the smoke and the rapidity with which particles settled out therefrom. The losses sustained must run into considerable tonnages in the course of a year's operation, and while we have no figures to present with regard to this particular plant, we feel safe in saying that by analogy, and our experience in other operations of a like nature, the loss cannot be less than from 3 to 5 per cent. of the total value of the ore.

In such operations where the drying of ores becomes a necessary step, precautionary measures should be taken to recover the values that would otherwise be lost in this operation.

In a more recent case where a complex zinc-iron ore (sulphides and oxides mixed) was to be treated by a dry method, it became necessary to dry the material after it had been crushed to about 1.5-in. size, driving off a relatively small percentage of moisture contained therein. This moisture would vary from 2 to as high as 10 per cent., although more generally it was in the neighborhood of from 3 to 4 per cent.

The type of drier used was a rotary cylinder 30 ft. long, 4 ft. in diameter, set at a pitch of 0.25 in. to the foot. The inside wall of the shell was equipped with numerous lifting-angles which effected a frequent dropping of the ore through the current of hot gases, but which with a relatively dry material had the disadvantage of causing more abrasion and stirring up more dust than was necessary, with a consequent loss that far exceeded all expectations. That losses would be suffered, due to the drying-operation, especially since the draft was induced by a small exhaust-fan, was to be expected, and it was determined that before any dust-collecting installations were to be made, the actual loss sustained in this operation was first to be measured. As a precaution against an excess loss of the dust, the draft-fan was operated with just enough speed to carry away the gases from a very light fire. The fan was so accurately adjusted that a reduction of but 15 to 20 rev. per min. caused some of the furnace-gas to leak through the brick setting and other openings, into the mill.

We had here an ore that in its general make-up is representative of a very large class of material that engineers are frequently called upon to dry. The ore carried about 20 per cent. of zinc as its principal value. This zinc was mostly in the sulphide state, having some iron chemically combined with it, so that the individual crystals varied from a light semi-translucent brown to quite a dark brown, opaque in appearance. The iron associated with the zinc was present primarily in three forms; that is, the sulphide, specular hematite, and magnetite. The percentage of iron in the ore was about equal to that of the zinc. As to other values, there was a small quantity of lead as galena carrying a few ounces of silver to the ton. The gangue consisted primarily of silica and some lime. Some of the mineral values were oxidized, considerably further complicating the mixture, and there were present in appreciable quantities some carbonates of zinc, lead, iron, and manganese.

EXTENT AND EFFECT OF ABRASION.

The type of drier in use at this plant was faulty in at least one detail, that being the rather unfortunate location of the feed with respect to the gas-outlet. The feed-pipe discharged its ore into the drier-shell at just about the point where the velocity of the gases was greatest; hence the power to draw fine material into the stack even before it came in contact with the drier-shell was great. Even with ore carrying from 10 to 15 per cent. of moisture, a heavy loss was experienced, and with actually wet ore the conditions were not much improved, since it thus became necessary to drive harder to effect the same degree of dryness. Hence the velocity of the gases passing through the shell was sufficient to carry along particles of considerable size and the net results were much the same.

Since superficial investigations indicated a very heavy abrasion-factor in the drier, which in part resulted in heavy tonnage- and zinc-loss, it became desirable to have a reasonably accurate measure of this factor. A test to determine this was carried out as follows:

It was attempted to have the conditions during the test as nearly as possible identical with those existing during regular mill-operations, the only difference being that the escape of dust should be minimized to the greatest possible extent, so that as nearly as possible all the material fed to the drier would be accounted for in the products derived therefrom. Even though the ore was practically dry, the drier was thoroughly heated before the test began. During the time of the test, fire-doors and ash-pit doors were closed up tightly, and the fans, both force and draft, were stopped, so that the only draft passing through the machine was such as would be induced by the tendency for the hot air within the drier to rise through the gas-connections and fan-casing, and cause fresh cold air to be sucked in through the leaks and other openings.

Precaution was taken at the discharge end of the drier so that contamination between the ore on which the test was being made and the material piled up in the back connection could not take place.

A measure of the abrasion can be had only by comparing the screen-analysis of the material when fed to the drier with the screen-analysis of the same material as it issues therefrom. For the purpose of producing as definite results as possible, the crude ore to be fed to the drier consisted of pieces no larger than those passing through the Blake crusher, about 2-in. ring, while also everything finer than 16-mesh was eliminated. Thus a direct relationship is to be had, since any material finer than 16-mesh in the discharge from the drier must necessarily have been produced during the passage of the ore through the machine. The ore as it was discharged from the drier was again screened over 0.25-in. and 16-mesh screens, the same as the feed. The relationship between these screen-analyses directly measures the abrasion-factor.

During the time of the test the stack-draft, through the drier, carried out into the atmosphere a small proportion of very fine material, all of which was —200-mesh, and which represented the actual quantity not accounted for between the feed and the discharge. On account of the fineness of this dust, this must be charged to the —200-mesh material in the screen-analysis of the dried ore. The results of this test are as follows:

The test was made on 2,177 lb. less 0.8 per cent. of moisture, or 2,160 lb. of dry ore, of which

83.7 per cent. was — 2-in. ring + 0.25-in. mesh.
16.3 per cent. was — 0.25-in. mesh + 16-mesh.
0 per cent. was — 16-mesh.

The ore discharged from the drier was 2,110 lb., with negligible moisture-content, showing a loss of 50 lb. of ore during the test, or 2.3 per cent., which must be charged against the—200-mesh product. This loss was represented by the fine material carried away by the draft. he dry ore discharged showed as follows:

Dry Ore.	Pounds.	Per Cent.	Interpolated when Adding Loss to —16-Mesh Product. Per Cent.
+ 0.25 in., . . .	1,611.0	76.3	74.4
— 0.25 in. + 16-mesh, .	288.5	13.7	13.3
— 16-mesh, . . .	210.5	10.0	12.3
	2,110.0	100.0	100.0

From the above data it will be noted that 12.3 per cent. of the total ore was ground into sizes finer than 16 mesh simply by the tumbling action taking place in the drier.

A screen-analysis of the —16-mesh material in the drier product and interpolated with the loss charged to —200-mesh float, as well as this screen-analysis applied to the total dry ore, shows:

Screen-Analysis on —16-Mesh Material of Dried Ore with Loss of 2.3 Per Cent. Added to Float. Per Cent.		Screen-Analysis of Total Ore After Passing Drier. Per Cent.
	+ 0.25 in.	74.400
	— 0.25 in. + 16 mesh.	13.300
11.74	— 16 + 20	1.420
10.17	— 20 + 24	1.240
8.00	— 24 + 28	0.970
5.34	— 28 + 34	0.650
8.00	— 34 + 42	0.970
6.44	— 42 + 50	0.790
6.53	— 50 + 62	0.797
5.57	— 62 + 74	0.695
4.78	— 74 + 86	0.585
9.30	— 86 + 109	1.146
3.40	— 109 + 125	0.422
4.40	— 125 + 150	0.538
7.03	— 150 + 200	0.862
6.86	— 200 grit	0.838
2.44	— 200 float	0.293

NOTE :—The —200-mesh material was treated in an up-cast, dividing the product into grit and float, the former representing a good table-feed, while the latter consisted of real slime, difficult, if not impossible, to treat mechanically.

From the above screen-analysis it will be seen that the drier acts as a tumbling-mill, reducing a considerable proportion of the ore, relieving the crushing-department of 12.3 per cent. of the work it would otherwise be called upon to perform. On the face of things, in the event of the abraded material being recovered and not lost with the drier-gases, this action might appear to be advantageous. The abrading action of the drier, however, throws at least 0.3 per cent. of the total ore into a product which becomes practically useless from a separating point of view. Since in the test all the original fines had been eliminated from the feed, it is likely that the quantity of —200 float in the drier-discharge is more nearly 1 per cent. on the total ore. By weight this is not so serious, but the assay of these products shows that the heaviest abrading effect takes place on the zinc. It was found that with the crude ore assaying 19.1 per cent. of zinc, the abraded or —16-mesh portion of the dried product as discharged from the drier showed an assay-value of 29.4 per cent. of zinc. The —200-mesh material contained in the dry ore shows a zinc-value of 26.1 per cent. We have then as follows:

Pounds. Per Cent.
2,160 @ 19.1 = 412.5 lb. of zinc in ore.
2,160 @ 12.3 = 266 lb. of ore-abraded material.
266 @ 29.4 = 78.2 lb. of zinc in abraded material.

$$\frac{78.2}{412.5} = 18.9 \text{ per cent. of zinc abrasion.}$$

From this, the abrading effect, while the ore is passing through the drier, is 50 per cent. greater on the zinc mineral than it is on the total ore, proclaiming the zinc as the greatest sufferer.

In the event of the dust thus made being lost, the abrading effect of the drier becomes very serious, while if the dust is recovered and disposed of in the proper places in the system, this effect, while not to be particularly desired, does not represent so disadvantageous a factor.

The effect of the drier-abrasion will be pointed out more in detail in recording a test on the entire reduction and dust-collecting portion of the plant in question, but at this point it may be well to point out three ways in which the drier-abrasion may be very materially reduced.

(1) Reducing the number of lifting-angles in the shell, which will cut down the number of times the ore is dropped.

(2) Increasing the pitch of the shell to 0.5 in. per foot, from 0.25 in. This will cause the ore to travel through the drier at practically double the speed.

(3) Introducing both No. 1 and No. 2 methods simultaneously.

The increased pitch of the drier-shell would result in a larger capacity, however, at the expense of the efficiency of the machine when moisture in the ore runs high.

RECOVERY OF DUST-LOSSES.

This particular plant, in which an entirely dry process was being operated, was equipped with an extensive dust-system operated by two 45-in. steel-plate exhausters, which, by suitable connections and draft-gates, caused a slight vacuum in the housing of all crushing-machines, as well as at all points where ore was handled, such as in the boots and heads of elevators, chutes, bins, and feeders. In this manner, with a properly adjusted suction-system, any dry-plant can be kept effectively clear of the obnoxious dust, with the result that instead of causing injury to attendants (by either the danger of lead-poisoning or phthisis), or damage to shafting, bearings, pulleys, and belting, this product, which is in most instances a valuable one, is collected and subject to treatment; thus, instead of suffering the losses as pointed out, at least a large portion of them may be recovered. The dust collected throughout the mill was discharged into a centrifugal dust-collector, Fig. 1, which was at first operated dry, but its efficiency was not as high as was desired. The velocity of the air was somewhat greater than that which would have permitted practically all of the dust to settle out, so that the apparatus under such conditions worked as a very excellent separator, inasmuch as the collected product was of a gritty texture, even though practically all of it would pass through a 200-mesh screen. Only the very finest of the dust passed out through the stack of the collector, and this was almost pigment-like in physical characteristics, resulting in no gritty feeling whatever, when rubbed between the fingers.

This type of dust-collector we have used with considerable success in several cases where very much larger volumes of gas were being handled, and found that to improve the efficiency the introduction of very fine water-sprays or steam-jets was most effective. The steam when condensing, or a very finely divided water-spray, acts as a powerful scrubbing-agent, wetting the particles of dust, thus enabling them to be caught by the film of water on the sides of the collector when the centrifugal force causes them to impinge against it.

To improve the efficiency of this dust-collector, both steam and water were introduced, using for this purpose a portion of the exhaust steam from the main engine. This was delivered into the inflow-pipe and in the direction of the flow of the air, while three jets of water divided into very fine sprays by means of an ordinary garden-hose nozzle were fitted into the sides of the collector about on a level with the center of the inflow-pipe. The bottom of the collector was sealed by about 6 in. of water, and all of the product, consisting of a thin sludge, was collected in a settling-tank below, from which the excess water was allowed to run to waste. The effect of this system was astounding, inasmuch as the collection of the dust was practically perfect, the air issuing from the top of the collector being barely discolored by the presence of any solid particles, and all efforts at collecting such escaping dust were unsuccessful, even by filtering through cotton. The tank collecting the slimes acted as a thickener, allowing the excess water to overflow, from which a slime-table feed could be drawn, while the excess water could either be returned to the spray-nozzles or be permitted to flow to waste.

For the purpose of collecting the drier-dust losses, the draft-fan of the drier was by-passed and the gas-outlet equipped with a blast-gate and connected with a large chamber, from which the exhausters received their dust-laden air. All of the dust-pipe connections throughout the plant lead to this central chamber, which for lack of a better term may be called "the vacuum-box," and which, due to its volume and the consequent reduction in velocity of the air or gas passing through it, permitted some of the heavier dust to settle.

TEST OF DUST-COLLECTING SYSTEM.

With provision thus made for collecting not only the relatively small quantity of mill-dust but also the large amount of ore lost in the drier-operations, a complete test of the drying- and crushing-plant with dust-collecting system in operation was carried out. This test gave some rather remarkable and interesting data, and fully demonstrated, not only the extent to which milling-operations may suffer, due to a loss of the drier- and mill-dust, but also the value of a simple and efficient dust-collecting system. The draft was again regulated so as just to prevent the escape of any furnace-smoke into the mill.

Since there was no longer any appreciable escape of valuable dust possible, such material as was not accounted for by the feed to the table-department could only distribute itself in three different products: *i. e.*, vacuum-box, dust-pipes, and slime-box. These three products were carefully collected and weighed after the test, everything having been thoroughly cleaned before the test began. These, as also the concentrator-feed, were carefully sampled, screen-analyzed, and assayed, so that a full and complete distribution of the zinc in the various products could be arrived at, as well as an ultimate screen-analysis of the total ore.

In the screen-analyses there is recorded in each case a —200 grit and a —200 float-product. These figures were obtained by agitating the —200-mesh material in each case in a beaker, while at the same time a small stream of water was allowed to flow into the beaker, thus overflowing, and carrying with it the finer particles of the —200-mesh product. In this manner we divided this product into what might properly be termed "grit" and "slime." However, the action was more intense than it would be in a settling-apparatus, and hence shows a slightly higher proportion of float material than actually exists in the crushed ore.

Seven thousand pound of crude ore was weighed out and fed through the Blake crusher to the drier, which crude ore contained 0.7 per cent. of moisture, accounting for about 50 lb., which, however, is not taken into account in the following figures, since the factor of error, in weights, collection of products, and observation, is probably at least as large as this moisture-factor.

This 7,000 lb. of ore was distributed in the four products as follows:

	Pounds.	Per Cent.
Concentrator-feed,	5,861	83.6
Vacuum-box,	601	8.6
Suction-pipes,	24	0.34
Slime-box,	514	7.34

Table I. shows the distribution of the ore, by weight, in the various sizes in each of the four products, and the assay-value of zinc in each of the products resulting from the screen-analysis.

TABLE I.—*Percentage-Weight and Value of Ore in Various Sizes of Each Product.*

	Concentrator-Feed.		Vacuum-Box.		Suction-Pipes.		Slime-Box.	
	Weight	Zn.	Weight.	Zn.	Weight.	Zn.	Weight.	Zn.
	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent
— 16 + 20....	19	14.1						
— 20 + 24....	11	17.3						
— 24 + 28....	9.50	17.9						
— 28 + 34....	6.25	19.6						
— 34 + 42....	8.25	19.2	0.50	10.5				
— 42 + 50....	7 —	19.1	1.75	15.3				
— 50 + 62....	6 —	20.6	3.75	19				
— 62 + 74....	4.75	21.8	7.25	20.8				
— 74 + 86....	3 —	21.3	10.75	22.4				
— 86 + 109....	6.50	20.6	25.—	26.6				
— 109 + 125....	2.50	20.1	8	27			7.—	18.5
— 125 + 150....	2.50	20.4	8	25.7	5	20.2	5	21.5
— 150 + 200....	3.50	21.8	14.50	23.8	8	20.5	12.5	22.8
— 200 Grit.......	6.35	20.9	11.27	27.4	57.85	19.35	11.1	21.-
— 200 Float.....	4.65	17.1	9.23	21.3	29.15	18.50	64.4	19.6

In order that a complete screen-analysis of the combined products, as well as a distribution of the actual zinc-content of the ore in the various products, may be obtained, Table II. was prepared, showing actual distribution by weight of the material and the zinc-content, from which is calculated the average zinc-analysis of each of the four products made.

TABLE II.—*Actual Weight and Values of Ore in Various Sizes of Each Product.*

	Concentrator-Feed.		Vacuum-Box.		Suction-Pipes.		Slime-Box.		Total.	
	Pounds.	Zn.	Pounds.	Zn.	Pounds.	Zn.	Pounds.	Zn.	Pounds Ore.	Pounds Zn.
+ 20.......	1,112	156							1,112	156
+ 24.......	645	111.5							645	111.50
+ 28.......	551	98.6							551	98.60
+ 34.......	363	71							363	71
+ 42.......	480	92	3	0.35					483	92.35
+ 50.......	410	78.3	10.5	1.61					420.5	79.91
+ 60.......	351	72.2	22.5	4.27					373.5	76.47
+ 74.......	270	58.8	43.5	9.05					313.5	67.85
+ 86.......	176	37.5	64.6	14.47					241.6	51.97
+ 109.......	375	77.2	150.4	39.90					525.4	117.10
+ 125.......	141	28.3	48.—	12.95			36	6.65	225	47.90
+ 150.......	141	28.7	48.—	12.34	1.20	0.252	25.70	5.52	215.9	46.81
+ 200.......	199	43.3	87.1	20.72	1.92	0.403	64.20	14.63	352.22	79.05
— 200 Gr.	372	77.8	67.8	18.54	13.89	2.685	57.—	11.95	510.69	110.97
— 200 Fl.	273	46.7	55.4	11.81	6.96	1.275	331	64.85	666.36	124.63
									6.998.67	1,332.11

The average of the heads is 19.01 per cent. of zinc by calculation. From the above data we then have the zinc-distribution in the various products as follows:

	Pounds.	Per Cent.	Zinc Distribution. Pounds Zinc.	Zinc Distribution. Per Cent.
Concentrator-feed,	5,861 @	18.4 =	1,077.5	81
Vacuum-box,	601 @	24.3 =	146	10.93
Suction-pipes,	24 @	19.23 =	4.62	0.35
Slime-box,	514 @	20.15 =	102.9	7.73
			1,331.02	
Crude ore,	7,000 @	19.01 =	1,330.7	100

The reasonable correctness of these data is proved by the assay-value of each of these products on the original samples, from which we have to note the following comparison:

Product.	Assay from Sample. Per Cent. of Zinc.	Assay by Calculation. Per Cent. of Zinc.
Sizer-feed,	18.7	18.4
Vacuum-box,	24.7	24.3
Suction-pipes,	19.4	19.23
Slime-box,	20.3	20.15

From the above data it is possible to set up a combined screen-analysis as well as the zinc-distribution, and also the calculated zinc-assay of each of the products in the combined screen-analysis, as shown in Table III.

TABLE III.—*Combined Screen-Analysis and Zinc-Distribution.*

Size.	Quantity by Weight in Products.	Quantity of Total Zinc in Each Product.	Zinc-Assay of Each Product.
	Per Cent.	Per Cent.	Per Cent.
— 16 + 20	15.9	11.72	14.1
— 20 + 24	9.22	8.38	17.3
— 24 + 28	7.86	7.40	17.9
— 28 + 34	5.18	5.33	19.6
— 34 + 42	6.90	6.93	19.15
— 42 + 50	6.01	6.—	19.03
— 50 + 60	5.33	5.73	20.46
— 60 + 74	4.48	5.10	21.67
— 74 + 86	3.44	3.90	21.60
— 86 + 109	7.51	8.79	22.30
— 109 + 125	3.21	3.60	21.32
— 125 + 150	3.08	3.52	21.70
— 150 + 200	5.03	5.93	22.47
— 200 Grit	7.28	8.33	21.72
— 200 Float	9.52	9.36	18.71

A comparison of the actual weight of zinc in each of the screen-products of the calculated combined screen-analysis of all of the products, and the actual weight of zinc contained in the screen-products of the vacuum-box, suction-pipe, and slime-box material taken together, gives the percentage of total zinc which did not directly reach the concentrating-department, but which was withdrawn during the drying- and crushing-operations, and collected separately in the three products mentioned.

Concentrator-Feed.	Pounds Zinc.	Loss. Pounds Zinc.	Per Cent.
— 16 + 20	156		
— 20 + 24	111.5		
— 24 + 28	98.6		
— 28 + 34	71		
— 34 + 42	92.35	0.35	0.38
— 42 + 50	79.91	1.61	2.02
— 50 + 62	76.47	4.27	5.58
— 62 + 74	67.85	9.05	13.33
— 74 + 83	51.97	14.47	27.88
— 86 + 109	117.10	39.90	34.10
—109 + 125	47.90	19.60	40.80
—125 + 150	46.81	18.11	38.70
—150 + 200	79.05	35.75	45.20
—200	235.60	111.10	47.20

Of the total ore fed to the drier, but 83.0 per cent. by weight directly reached the concentrating-department, containing but 81 per cent. of the total zinc in the ore. On the whole, there is no material effect in the direction of concentration by the drying and crushing action, but individually the three separately-collected products show marked tendencies in this respect. Particularly is this true of the vacuum-box product, in which a relatively larger proportion of the total zinc was recovered than of the ore, showing the zinc somewhat concentrated in this material. The suction-pipe and slime-box products do not greatly vary from the assay-value of the original ore.

Drier-dust losses, as well as such products as are withdrawn from the mill by means of the suction-system, are usually charged against slimes; and in the event of a complex ore are frequently considered as having been fortunately disposed of, since further treatment is difficult, if indeed, in some cases, not quite impossible. That such losses cannot be charged against

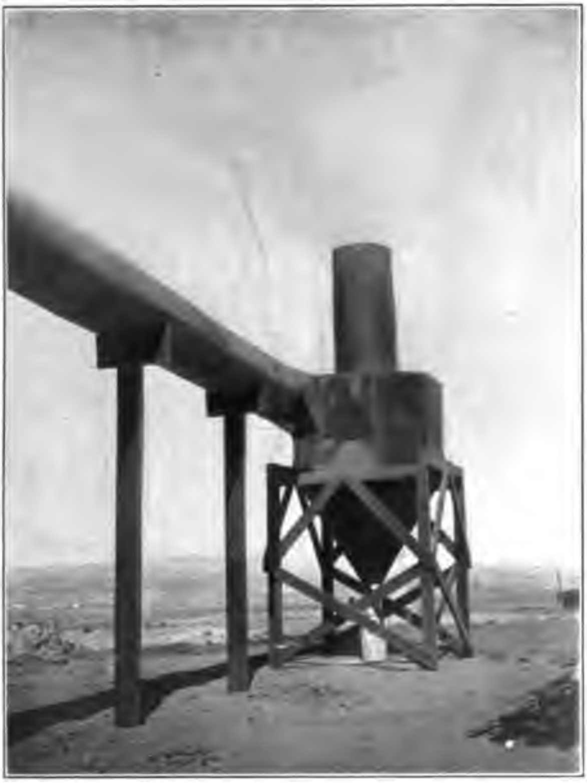

FIG. 1.—DUST-COLLECTOR IN DRY OPERATION. NOTE THE SLIGHT DISCOLORATION OF THE OUTLET AIR AT TOP OF STACK.

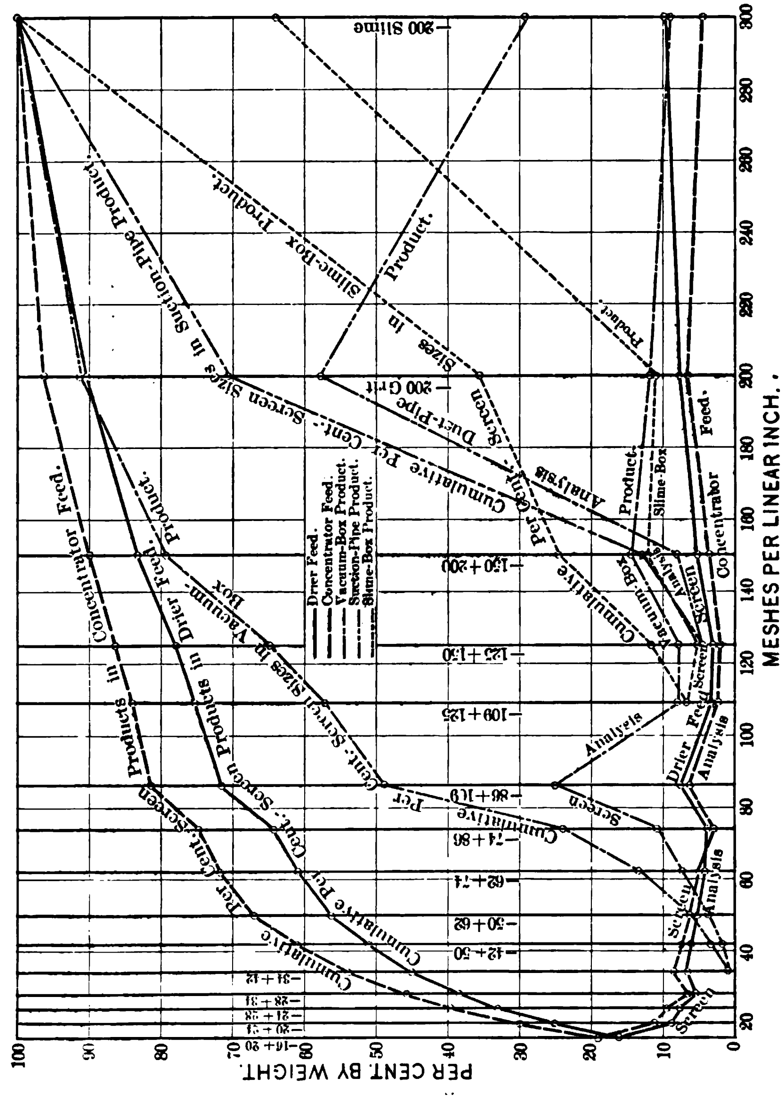

FIG. 2.—CURVES OF SCREEN-ANALYSES OF DRIER-FEED AND THE FOUR RESULTANT PRODUCTS.

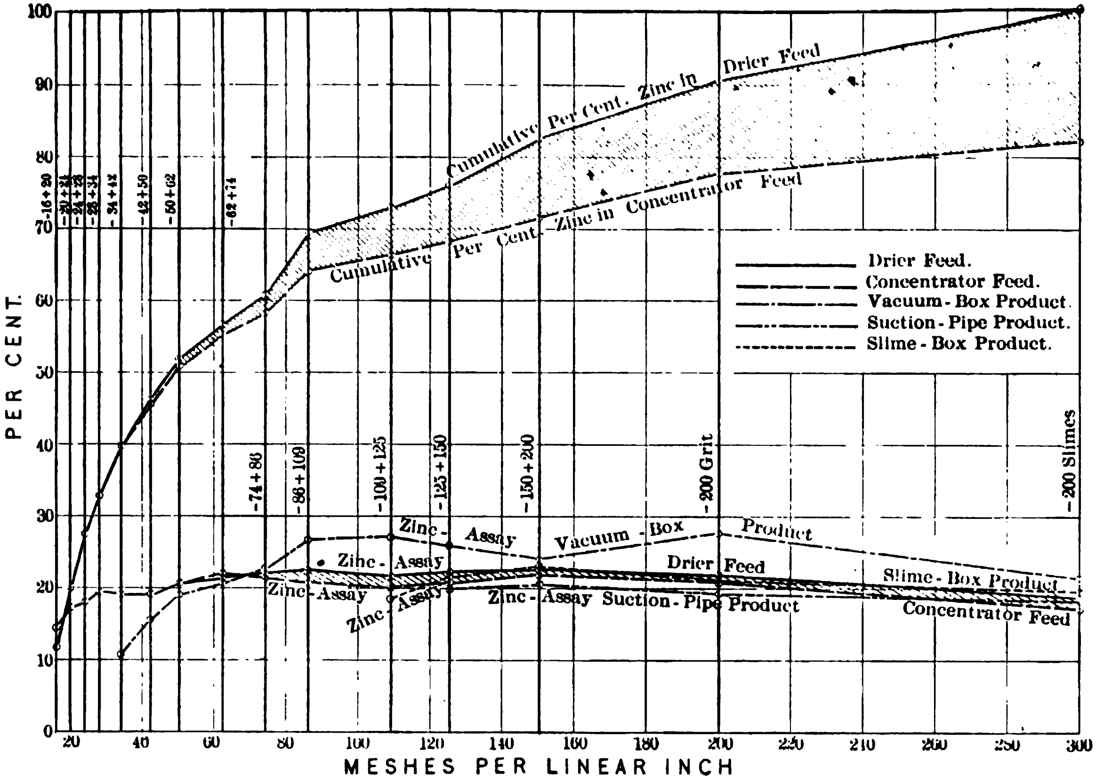

FIG. 3.—CURVES OF ZINC-ASSAYS OF SCREEN SIZES IN THE DRIER-FEED AND THE FOUR RESULTANT PRODUCTS.

the slimes, is demonstrated by the screen-analyses of the various products, which are graphically represented in Fig. 2. The legend identifies the curves, in addition to which each curve is captioned. The screen-analysis of each product is shown, as well as a curve showing the screen-analysis in cumulative percentage. Certain conditions as revealed by these curves deserve special attention. The difference in the screen-analysis between the crude ore fed to the drier and that going to the concentrating-department, as shown by the length of the ordinates between these two curves, is to be especially noted. These two curves cross at a point represented by a screen of about 65-mesh, the coarser products being in larger proportion in the screen-analysis of the concentrator-feed, while the products finer than 65-mesh are heavier in the crude ore, due, of course, to the suction-action caused primarily by the drier-draft, assisted to a small extent by the general dust-suction system throughout the plant. A distinct gradation, by size, is noted through the various products to the slime-box material, the latter showing by far the largest proportion of —200-mesh material. Each of these screen-analyses is also represented by a cumulative percentage-curve, which shows the relative fineness of the product it represents by the rapidity with which it rises as the sizes become finer. Thus the curve representing the cumulative percentage of the screen-products in the concentrator-feed is the most nearly horizontal, while the curve representing the cumulative percentage of the screen-analysis of the slime-box product is the most nearly vertical.

The most important collected product to be dealt with is that which settled in the vacuum-box, and which by its screen-analysis shows but 20.5 per cent. finer than 200-mesh, of which about one-half is still of a gritty character, and represents a good concentrator-feed. That the minimum draft of this particular drier is sufficient to cause very serious losses is evidenced by the fact that the vacuum-box product contains particles as coarse as 34-mesh, and a very heavy percentage of this total product lies between 80- and 100-mesh screen. This material is in excellent form for further treatment, and, if permitted to escape into the atmosphere with the drier-smoke, entirely aside from the fact that it carries concentrated zinc-values, would spell a loss which no milling-operation can suffer without very seriously impairing its commercial possibility.

In Fig. 3 are shown graphically the zinc-assay values of the screen-sizes of all five products under consideration, as well as two curves giving the cumulative percentage of the zinc in the various screen-sizes of the drier-feed and the ore which directly reaches the concentrating-department. The sectioned area between these two cumulative percentage-curves directly represents the percentage of total zinc lost as the grades grow finer, resulting in a total loss of 19 per cent. of the zinc by the drying- and crushing-operations. The latter, however, are only a very small factor in the total.

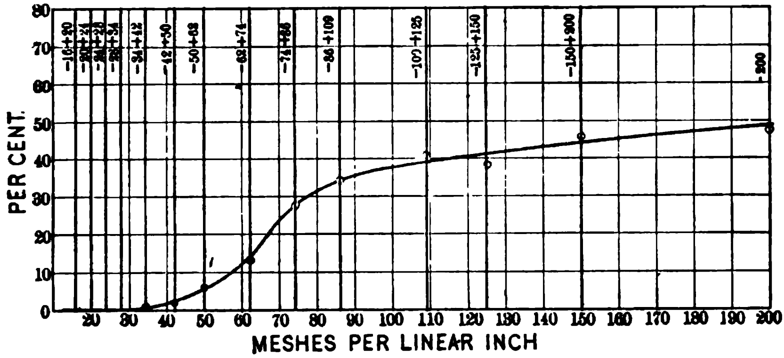

FIG. 4.—CURVE SHOWING PER CENT. OF ZINC-CONTENT WITHDRAWN DURING THE DRYING- AND CRUSHING-OPERATIONS, BASED ON THE TOTAL ORE.

The narrow sectioned area between the curves representing the zinc-assays of the screen-sizes in the drier-feed, as against that going to the concentrating-department, shows the loss in assay-value, due to the concentrating effect caused by the draft, which in turn is represented by the higher zinc-assay value of the screen-sizes from the vacuum-box product. The slime-box product shows much the same value as the crude ore, while the zinc-content of the suction-pipe product is somewhat lower.

In Fig. 4 is shown a curve which represents the percentage of the zinc in each screen-product of the total ore withdrawn during the drying- and crushing-operations. It is to be noted that this action does not begin on particles coarser than a 34-mesh screen, but from this point to 86-mesh the loss increases rapidly, after which the increase in loss is more gradual, still rising, however, until on the —200-mesh material a total loss of 47.2 per cent. of the zinc is suffered.

The fact that such a relatively large percentage of the material withdrawn from the drier by the draft is not in slime condition, is very satisfactory, since it permits of collecting this dust in a simple manner, it to be returned to the system just before reaching the concentrating-department. Thus a material percentage of the total ore by-passes the crushing-department.

The economical collection of these fugitive values is already demonstrated and can be made a part of every dry-milling operation at very little installation-cost, while the operating-cost is so low as to be almost negligible in comparison with the benefits derived.

[TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS.]

A New Electric Miners' Lamp.

BY D. B. RUSHMORE, SCHENECTADY, N. Y.

(New York Meeting, February, 1912.)

TORCHES were used by the early Romans for mine-lighting, and these were followed by open lamps or earthen jars filled with tallow or oil, and later by candles. In early coal-mining, explosive gases seldom occurred, and, if they were encountered, the danger of explosion was materially diminished by the absence of any ventilating-system and the consequent insufficient circulation of air to form with the methane an explosive mixture.

As the art advanced, the danger of fire-damp became more apparent, and methods for testing the air were resorted to as early as in the fifteenth century. The method of testing consisted in lowering a dog in a basket down the shaft. As soon as he encountered fire-damp he would commence to howl, after which he was withdrawn, and a leafy bush was fastened to the end of the rope and run rapidly up and down the shaft. This so disturbed the accumulation of gas as to cause it to rise out of the shaft. Later this odd method of testing was improved upon through the discovery that a light inserted in a body of fire-damp would produce a blue flame. Candles were used for this purpose, and, when fire-damp was thus discovered, brushing or some other primitive method of fanning was resorted to for driving out the gas. In some instances the gas was burned away, regardless of the danger thereby incurred. A man, wearing a well-moistened coat and a mask with glass spectacles, descended in the mine a few hours before each shift. With a long stick, to which was fastened a burning torch or candle, he would crawl on the floor with the stick so raised that the light would sweep the roof and ignite the gas which it encountered. In this position the flame of the explosion would pass over him, after which he would retreat from the gases resulting from the combustion. The occurrence of serious accidents resulted in the substitution of a method by which the candle was moved along the headings by means of an endless rope, which was supported on pulleys near the roof and could be operated from a safe point.

In 1815 Davy invented the safety-lamp which bears his name and the principle of which is embodied in the numerous safety-lamps of later inventors. Although many improvements have been made in the safety-lamp, it is still far from perfect; and the present indication is, that, in order to obtain a perfectly safe and reliable lamp, the electric light must be resorted to.

In metal-mines, and even in coal-mines where the gases contained in the atmosphere are sufficiently diluted to render them harmless, illumination may be had by any form of naked light. But in collieries where considerable quantities of fire-damp and coal-dust are liable to collect, the lamp-flame must be protected from direct contact with the gas, and it becomes necessary to use some sort of safety-lamp.

The most common of all miners' lamps is the small tin lamp which generally is carried on the miner's cap. A cotton wick extends through the spout into the fluid, which is generally some kind of petroleum-product. While this lamp, when trimmed and kept in order, gives a fairly-good light, the objections to its use are many. The almost continuous smoking seriously impairs the purity of the air and thus handicaps the miner in his work. It is easily blown out by blasts and drafts. The leakage of oil from the lamp is a constant annoyance, and the time required for refilling and trimming the lamp is considerable.

The acetylene lamp, used to a considerable extent in a number of mines, is a marked improvement on the oil-lamp. Not only is the illumination materially better, but the smoking and leakage of the oil-lamp are practically eliminated. The difficulties of handling seem, however, to have prevented its general adoption.

In mines liable to the intrusion of excessive amounts of fire-damp, safety-lamps must be used, either exclusively, or for purposes of inspection, preliminary to the removal of the gas. Several types of safety-lamps of more or less ingenious design have been placed on the market and are in actual use in the various gaseous coal-mines.

The Davy is an ordinary oil-lamp, the flame of which is surrounded by a cylinder of wire gauze, with a top of the same material. The principle of this lamp is that the cooling effect of the wire gauze prevents the propagation of the flame from the inside of the lamp to the outside atmosphere. After the lamp has been filled with oil and lighted, it is locked, so as to prevent the miner from having access to the flame, the wick of which is trimmed by a wire passing up through a close-fitting tube from the bottom.

The Clanny lamp is similar to the Davy, with the exception that the lower portion of the wire-gauze cylinder is replaced by a short glass cylinder, giving a somewhat better illumination.

Stephenson's lamp has a long cylinder of glass surrounded by wire gauze, and bonneted above by perforated copper. The air-feed is also through the gauze, going underneath and into the cylinder to the flame, thence out of the top as usual. This keeps both cylinder and gauze cool, and its relative security rests essentially on the regularity of the draft, for if the inside air becomes overheated the light goes out; so it must be suspended properly.

The Marsaut lamp is an improvement upon the Stephenson type, and can safely stand a fair amount of tilting. A great difficulty is, however, experienced in relighting it, and from the winding path pursued by the feed-air, proper circulation does not take place until the lamp gets hot.

The Hepplewite-Gray lamp admits the air at the top, down four tubes, and through an annular chamber above the oil-vessel. The only gauze employed is that covering the outlet and the annular inner chamber. A serious difficulty with this lamp is its liability to be extinguished when suddenly lowered.

In the Dick port-hole lamp, the air enters the lamp above the case, passes through the gauze, then descends to the flame, while the products of combustion rise inside the lamp, to be emitted through circular holes at the top of the bonnet. The bonnet is made of a seamless steel tube, and is light and strong.

The Wolf benzine safety-lamp[1] is a departure from the above safety-lamps in that: (1) it burns benzine or naphtha; (2) it contains a self-igniter which permits the relighting of the lamp

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[1] *Trans.*, xiii., 129 to 132 (1884–85); xiv., 410 to 418 (1885–86).

50 times without opening; and (3) it contains a locking-device which cannot be opened except by the use of an exceedingly powerful magnet.

While the safety-lamp possesses great value as a gas-detector, it is open to several serious objections. The first is the possibility that the gas, entering the lamp, and burning inside the cylinder, may overheat the gauze, which will then permit the flame to pass through it. Several types of lamps, however, are arranged to be self-extinguishing in case of ignition of gas inside the gauze. This is, of course, objectionable, as it will leave the men in darkness when such a danger occurs. The temptation of the miners to relight extinguished lamps, or open them for lighting pipes, etc., is very great, and has in many instances resulted in serious explosions. Finally, the illuminating-power of the lamps is very low.

For the above reasons safety-lamps of the ordinary type are being replaced in many localities, especially in bituminous coal-mines, by portable incandescent electric lamps. This type offers numerous advantages: increased safety in explosive atmospheres; increased illumination, ability to concentrate the light on a certain area, reliability and simplicity, elimination of smoke, cleanliness, smaller fire-risk, etc.

An objection often raised against the electric lamp is that it offers no means of indicating the presence of fire-damp. Several devices have been proposed, such as providing spiral wires coated with salts of platinum, etc.; but there seems to be at present no really satisfactory solution of this difficulty, although there is no reason why a solution should not be expected. To attempt to find a satisfactory fire-damp indicator is, however, merely to delay the general introduction of electric lamps; and there seems to be no real necessity for such an addition to a miner's lamp. The fire-damp detector might well be an entirely separate instrument, which could be installed in places where the fire-damp is expected to form first, such as near the roof, or in cavities therein. There is no reason why one or more gauze-lamps should not be supplied with each batch of electric lamps, and hung in the working-place where they can be easily seen, and where any outburst of gas would be most likely to occur.

Numerous electric mining-lamps have been designed, but few have proved efficient and durable. The design in the United States and abroad has followed very different lines. In Europe the hand-lantern has been almost exclusively considered, while the tendency here has been towards the perfecting of a combined hand-lantern and head-light. This is the logical result of the fact that the most satisfactory working-results can be obtained by a head-light, whose reflector concentrates and throws the light in the direction in which the miner works.

A new electric miners' lamp of improved design has recently been developed by the General Electric Co. to meet the increasing demand for a safe and reliable lamp. The idea of a systematic investigation as to the possibilities of developing such a lamp originated with W. J. Richards, vice-president and general manager of the Philadelphia & Reading Coal & Iron Co., to whom credit is due for his very active assistance.

The lamp is of the combination hand-lantern and head-light type, and consists of a miniature tungsten lamp-unit, operated from a light storage-battery. The lamp is provided with an efficient reflector, designed to illuminate a 9-ft. circle at a distance of 4 ft. from the lamp, and photometric records show about 2.5 or 3 candle-power at a distance of 2 ft. The battery has a capacity of 5 ampere-hours, and is of sufficient size to operate a lamp from 12 to 14 hours.

When used as a head-light, the lamp, with its reflector, is fastened to the miner's cap, and connected by means of a flexible cord to the battery, which is carried on a belt or from shoulder-straps, as shown in Fig. 1. When used as a hand-lantern, Fig. 2, the lamp-socket is removed from the cap-receptacle and inserted into the receptacle on the side of the battery, simply taking the place of the cable-attaching plug.

Some of the important points in connection with this design are: substantial construction; light weight and compactness; complete inclosure of all conducting parts; use of an absorbent for the electrolyte, to prevent spilling; small number of movable contacts; increased efficiency of a 4-volt lamp; efficiency of the reflector; impossibility of charging with reversed polarity; instant convertibility from hat-lamp to hand-lantern. These features will be considered in the order in which they have been mentioned.

Special care has been taken to select all materials used in the construction to withstand the hard service to which this lamp may be subjected. The steel battery-case is drawn from a single plate and is, therefore, seamless. The battery-jar is not of the brittle rubber commonly used, but has some elasticity left in it, so that, should the steel case receive a blow, there is little chance of the battery-jar being cracked. The cord from the battery to the lamp is metal-armored where the greatest wear and strain occur, but perfectly flexible where flexibility is required. The battery-case is locked with a magnetic lock which also locks the attaching-plug in the receptacle, thus preventing this plug from working loose and permitting the formation of a spark. The hat-lamp is sealed with a lead seal, which must be broken before the miniature lamp-globe can be removed.

This outfit has been designed, as already explained, for use as a head- as well as a hand-light, and therefore the weight of the battery, which must be carried on a belt, in the pocket, or by shoulder-straps, has been kept at a minimum consistent with strength and durability. The battery is flat and compact in order that it may lie as close as possible to the body. It weighs approximately 3 lb. and measures 6½ by 4⅝ by 1⅛ inches.

In order to prevent accidental short-circuits, causing sparks, all live parts are totally inclosed in molded compound. The battery-terminals, attaching-plug, hat-receptacle, and lamp-holder are all molded from hard fire- and acid-proof compound and thus perfectly protected from corrosion due to acid spray or gases.

An absorbent is used for separating the battery-plates and also filling up the extra space in the battery, except the mud-space. This absorbent acts like a sponge and holds the electrolyte in the battery even though it be inverted for some time.

Where low voltages are used, the pressure-drop caused by the contact-resistance may be serious. The movable contacts in this lamp have, therefore, been reduced to a minimum and have been so designed that they come together with considerable pressure.

The increased efficiency and life of the apparatus is principally due to the improved tungsten lamp. A 2-volt lamp is somewhat inefficient and has a comparatively short life. This

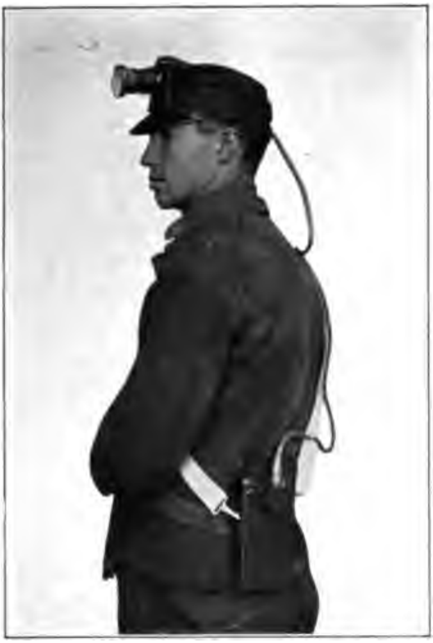


FIG. 1.—ELECTRIC MINERS' LAMP USED AS A HEAD-LIGHT.

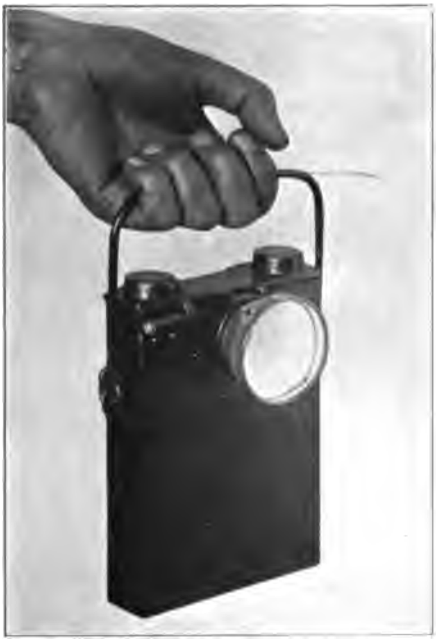


FIG. 2.—ELECTRIC MINERS' LAMP USED AS A HAND-LANTERN.

is due to the cooling effect of the large leading-in wires on the short filament. This cooling by conduction reduces the temperature of about one-third to one-half of the filament to a point where little or no light is produced. In order to produce the rated candle-power, the rest of the filament must be operating at a temperature far above a safe value, with a consequent unreliability and shortening of life. The filament of the 4-volt lamp, for which this outfit is designed, is about twice as long as a 2-volt filament, so that the percentage of length cooled by the leading-in wires in a 4-volt lamp is about one-half that in a 2-volt lamp. This allows the rest of the filament to be operated at a safe temperature, giving a reliable lamp of long life. The filament is, however, not so long as to cause mechanical weakness.

The reflector is made of stamped steel, enameled both inside and outside, thus absolutely protecting it from corrosion. Tests have shown that white porcelain enamel forms one of the best possible reflectors, and is very durable and easy to keep clean. The lamp-globe is protected from injury by a glass plate, fastened to the rim of the reflector by a spring. This glass further tends to prevent dust from accumulating on the inside of the reflector.

The contacts on the battery-terminal and plug are made concentric, so that, no matter how the plug is put in, the polarity must be the same each time. A separate plug is supplied for use in charging, and can be wired permanently on the charging-rack with its polarity correct. It will then be impossible to charge the battery from this plug with reversed polarity.

While the demand for an electric mine-lamp seems to be almost entirely for one to be worn on the cap, to supersede the present oil-lamp, it appeared very desirable to have an outfit that could quickly and readily be converted into a hand-lantern. This has been accomplished by designing the lamp so that, by a quarter-turn, it can be removed from the hat-receptacle and inserted, by a quarter-turn, into the plug-receptacle on the side of the battery.

[TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS.]

An Early Discovery of Fullers' Earth in Arkansas.

BY J. C. BRANNER, STANFORD UNIVERSITY, CAL.

(New York Meeting, February, 1912.)

DURING the past two or three years I have seen statements regarding the first discovery of fullers' earth in this country that seem to require correction or modification. One appears again in the *Mineral Resources of the United States for* 1909, part II., p. 738, where it is said that "fullers' earth was first discovered in the United States at Quincy, Fla., in 1893."

In this connection the following bit of unpublished history concerning fullers' earth in the State of Arkansas is worthy of attention.

In April, 1891, when I was State Geologist of Arkansas, a Swede named John Olsen, formerly a copper-miner in Michigan, but then living south of Little Rock, brought me a sample of what he thought was fullers' earth. He reported that it was from near the town of Alexander, the locality being 1 S. 13 W., section 8, SW. quarter of the SE. quarter. My chemist, Dr. R. N. Brackett, analyzed the material, and reported:

		Per Cent.
Silica,	SiO_2	63.19
Alumina,	Al_2O_3	18.76
Iron,	Fe_2O_3	7.05
Lime,	CaO	0.78
Magnesia,	MgO	1.68
Potash,	K_2O	0.21
Soda,	Na_2O	1.50
Loss on ignition,		7.57
		100.74

Water at 110° to 115° C. = 7.84.

Another sample from about the same locality, also brought by John Olsen, on analysis by Dr. Brackett gave:

		Per Cent.
Silica,	SiO_2	62.92
Alumina,	Al_2O_3	18.24
Iron,	Fe_2O_3	7.62
Lime,	CaO	0.76
Magnesia,	MgO	1.77
Potash,	K_2O	0.54
Soda,	Na_2O	1.44
Loss on ignition,		7.55
		100.84

Water at 110° to 115° C. = 6.95.

Analyses were also made of samples of imported fullers' earth, and the similarity between the domestic and the foreign materials was so close that it was thought worth while to have practical experiments made with the Arkansas earth.

On May 12, 1891, the results of these analyses were reported to A. M. Littlejohn, of Little Rock, Ark., who was at that time, as manager or otherwise, connected with the Southern Cotton Oil Co., which was using these clays for refining cotton-seed oil in its mills at Little Rock.

On Sept. 30, 1891, I went personally to Alexander and to Niemeyer and visited the region from which these earths had been reported. I collected a sample on 1 S. 13 W., section 8, SW. quarter of the SE. quarter, which, analyzed by Dr. Brackett, gave:

		Per Cent.
Silica,	SiO_2	64.38
Alumina,	Al_2O_3	17.29
Iron,	Fe_2O_3	8.25
Lime,	CaO	1.11
Magnesia,	MgO	0.80
Potash,	K_2O	1.41
Soda,	Na_2O	0.42
Loss on ignition,		6.95
		100.61

Water at 110° to 115° C. = 9.02.

It will be noted that these analyses are as nearly alike as could be expected. Of course, they were not, and could not be, regarded as settling the question whether the earths were or were not fullers' earths. But they were considered as quite favorable to such a theory, and Mr. Littlejohn was encouraged to give the earth in question a thorough practical test. For some time the Arkansas material was used in the oil-works.

Just how long it was used, I do not know, for Mr. Littlejohn, who had charge of the works, died in December, 1891, and I am unable to learn any of the details of his experience. Mr. Olsen, however, reports that the Southern Cotton Oil Co. finally gave up its use.

I recall the fact that it seemed to me, at the time when the experiments were being made, that the methods used in obtaining the earth must of a necessity lead to failure. This fullers' earth is the weathered portions of nearly horizontal beds of Tertiary clays that run through the low hills over a large area in the region S. and SW. of Little Rock. According to the miners' preconceived notion that ore must "improve with depth," it was expected that after the weathered edges of the clays had been passed the fullers' earth must likewise improve. I remember also that, just when Mr. Littlejohn thought the earth was going to be a great success, the miners began to congratulate themselves that the stuff was looking "good enough to eat," and consequently that it must be improving. Almost immediately, Mr. Littlejohn reported that it was not working well. To me the reason seemed very clear; the quarrymen had left the weathered zone behind, and had entered the unaltered clays.

I understand that the fullers' earth business in Arkansas is now well established and constantly growing. The only object of the present brief statement is to call attention to the fact that fullers' earth was known and was already being mined in the State of Arkansas as early as 1891.

Even since the present paper was written, an advance chapter of the *Mineral Resources for* 1910, upon the production of fullers' earth, repeats, at p. 4, the statement that the first "discovery of fullers' earth in this country" was at Quincy, Fla., in 1893.

SUBJECT TO REVISION.

[TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS.]

Contribution to the Study of the Pre-Cambrian Rocks of the Harney Peak District of South Dakota.

BY GORDON S. DUNCAN, A.R.S.M., A.I.M.M., LONDON, ENGLAND.

(New York Meeting, February, 1912.)

THE U. S. Geological Survey, I believe, has almost completed a study of the Harney Peak quadrangle, preliminary to the publication of a report on that district. As I was engaged for some months on an investigation of the mode of occurrence of the ore-bearing formation in this area, it has seemed to me timely to make public the results of my study, as a contribution to the consideration of this subject.

It seems scarcely necessary to describe the locality and general geology of the Black hills of South Dakota, for since 1846 many have contributed information concerning this region, some of the reports bearing such well-known names as those of Walter P. Jenney, Henry Newton, S. F. Emmons, and C. R. Van Hise, while we have only to glance at *Bulletin No. 4 of the South Dakota School of Mines*, in which Prof. C. C. O'Harra, of that institution, has given us a bibliography of contributions to the geography and geology of the section, to find that other well-known men have studied the district, and have given us the benefit of their researches.

The U. S. Geological Survey has made geological maps of nearly the whole of the Black hills, the only quadrangle yet to be completed being, as stated, the Harney Peak district. During the late summer and autumn of 1910, it was my lot to examine and report on a considerable area in this neighborhood, which once had been prospected, and in some cases exploited, for tin. This paper gives an account of certain geological phenomena, noted at that time, which greatly influenced my conclusions.

The predominating formation of the district is pre-Cambrian, represented by a highly-cleaved mica-schist, usually very garnetiferous, and sometimes hornblendic. Rising like an island from the surrounding schist is a large mass of granite, known as

Harney's Peak. Originally the whole of this area was overlain by the Potsdam conglomerate, Carboniferous white limestone, and representatives of the Jurassic and Cretaceous series, the escarpments of which are prominent features in the landscape surrounding the pre-Cambrian formation, and which must have represented many thousands of feet above the present horizon. Erosion has therefore been enormous where the pre-Cambrian rocks outcrop. As well as the main occurrence of granite, there are numerous outcrops of pegmatite, greisen, and quartz throughout the schist formation. These appear as veins or ledges, in some cases parallel with the cleavage of the schists in their strike, in other cases cutting across it, and thereby causing some observers to consider them to be true fissure-veins. It was to these ledges that my attention was principally directed during my investigation, since it was here that tin had been found.

In this district, cassiterite occurs in a rock containing usually two of the constituent minerals of granite (*i.e.*, in a quartz-mica aggregate known as "greisen"), and more rarely composed of quartz alone. The greisen itself occurs in two forms:

(1) *Massive*, occupying the whole of the vein, the constituents being themselves arranged in a markedly segregated manner, as illustrated in Fig. 1, which shows an ideal section of a greisen ledge. Here cassiterite occurs on either of the walls, in association with the mica, or else in the center, where the largest crystals of mica usually are formed. It is rare to find cassiterite in the portions of the vein composed almost entirely of quartz, although occasionally large crystals of the mineral are met with there. It is, however, a fairly general rule, that where large crystals of cassiterite are found, the mica is in large crystals also, which suggests that the presence of cassiterite is not due to subsequent deposition.

(2) *Granular*.—This form of greisen is a fairly homogeneous mixture of quartz and mica, and is associated with pegmatite, occurring in ledges of that rock in irregular pockets and shoots, usually few and far between, and generally at the contact of the pegmatite with the schist. It seems to contain the more cassiterite the more mica is present, and always in smaller crystals than in the massive greisen.

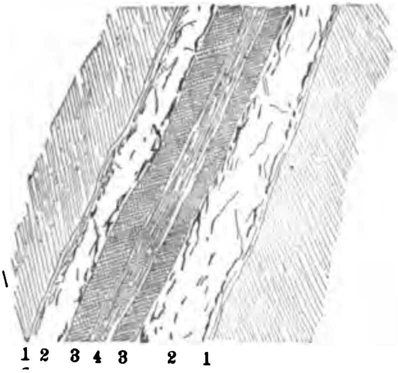
Fig. 2, a section of one of the ledges, is a good example of the manner in which granular greisen occurs in a pegmatite ledge.

I would note here that the presence of greisen, massive or granular, does not always indicate the presence of cassiterite, since I never noticed cassiterite in any rock where the three constituent minerals of granite were present or where disseminated tourmaline occurred in any quantity.

There are three types of ledges in this district:

(1) Those consisting of pegmatite containing erratic occurrences of granular greisen, usually from 3 to 20 ft. wide.

(2) Those consisting of massive greisen only.



1. Small crystals of mica near contact.
2. Quartz and a little mica disseminated through it in cracks.
3. Small mica crystals, becoming larger towards the center; very little quartz.
4. Much mica, very little quartz, and most of the cassiterite.

FIG. 1.—COWBOY LODE.

(3) Those consisting of quartz only, carrying little or no mica, practically no tin, and generally only a trace of gold, although in some cases gold in paying quantities has been discovered in large pockets or "bonanzas," distributed very irregularly throughout the ledge, as is usually the case with the white glassy quartz such as occurs in this region.

The relations between these ledges and the country schists and the main granite intrusion are two of the principal points to which I wish to draw attention.

The difference in texture, and to some extent in composition, between the granite batholith of Harney Peak and the ledges

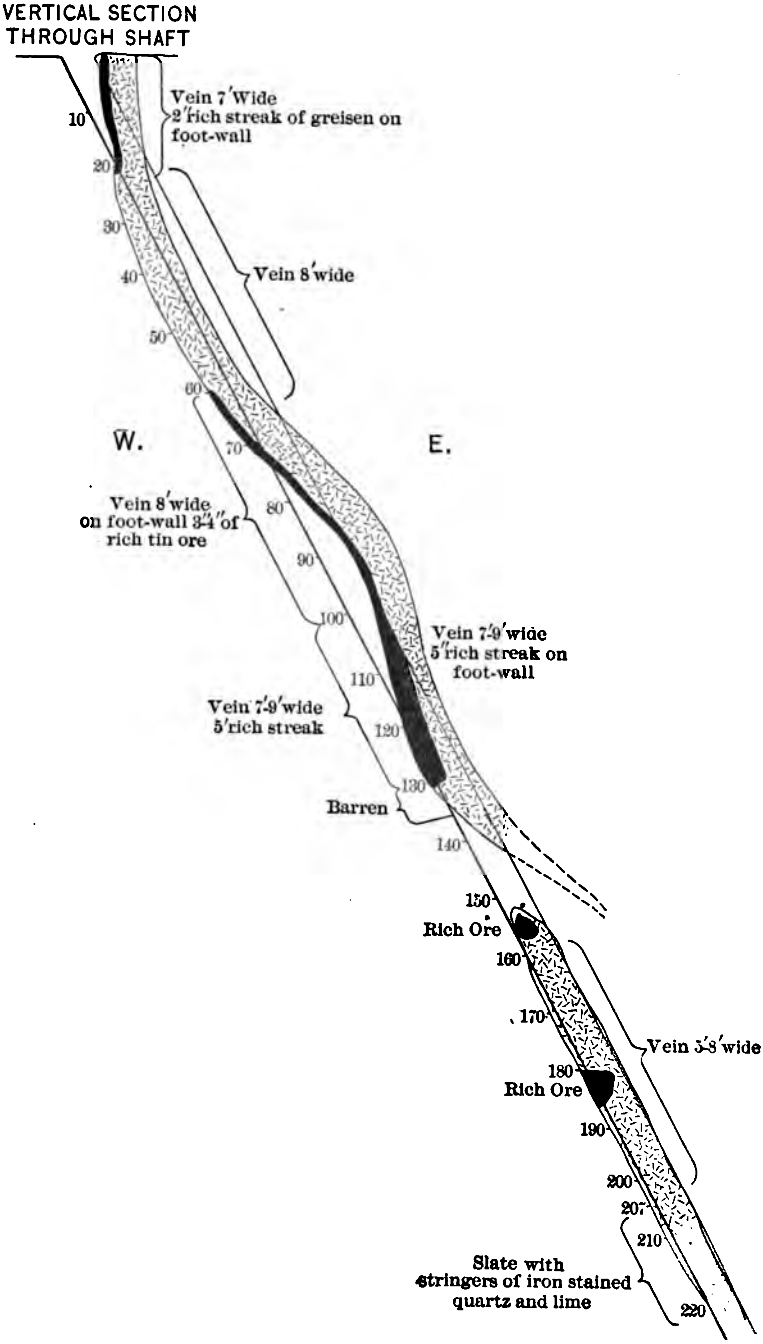


FIG. 2.—ADDIE MINE. (After T. Ulke.)

[4]

occurring in the vicinity is very marked. The granite of the former is for the most part of the coarse-grained normal variety, having the feldspar in white crystals, somewhat larger than those of the quartz and mica, and carrying much tourmaline of the usual deep blue variety. Never did I find in the granite any greisen, or any trace of cassiterite. Rarely does one see on Harney Peak any evidence of segregation of the minerals of the granite, beyond the fact that the feldspar is usually inclined to be idiomorphic and to occur in somewhat large crystals. The arrangement of the crystals shows in places a tendency to gneissic structure. Passing away from the central granite batholith, we find the outcrops of many granitic dikes, which show a somewhat marked structure in the arrangement of the crystals of the constituent minerals. The feldspars are all crystallized out in large idiomorphs; much tourmaline and a little columbite are present. Proceeding further westward, we pass outcrops of granite lenses, becoming very coarse-grained and showing a distinct foliation in their structure, almost resembling a pegmatite. Many outcropping lenses of quartz can be noticed. The mica-schist in this vicinity dips very steeply westward and masses of it stand out in high relief from the surrounding country, especially where they contain a core, as it were, formed by an included lense of quartz or pegmatitic granite. Continuing in a westerly direction, we find that the granite ledges begin to show marked segregation of their constituent minerals, and to present more of the texture of pegmatite. The quartz is inclined to be more segregated than its associates and to occur in veins and masses in the pegmatite, while now and again granular greisen is met with, sometimes showing crystals of cassiterite. The greisen, however, is very local in its occurrence, here as elsewhere, where it occurs in a pegmatite. A minute examination of one of these ledges would reveal a length of only a few feet (usually at the contact with the schist walls) in several hundred feet of the outcrop. As we proceed further westward, and away from the main granite intrusion, the quartz veins are more frequent, both in the schist and in the pegmatites, and at a distance of about 4 miles from Harney Peak veins or lenses of massive greisen occur. We see, therefore, from the foregoing, that the further away from Harney Peak, the more segregated, generally speaking, are the minerals, especially the quartz, in the pegmatites.

Southwest of Harney Peak, and in the Custer City district, we find a greater tendency than to the N. and NW. of the granite intrusion towards a more equal segregation of all the minerals of the pegmatites. The result is that large masses of quartz are less frequent in this district than they are at an equal distance from the granite intrusion at Hill City, although even here the quartz is inclined to segregate itself in the pegmatites. For example, the New York mica-mine, owned and operated by the Westinghouse Electrical & Manufacturing Co., has opened up a lense of pegmatite 30 ft. wide, which is distinctly more quartzose at the center than at the contact with the schist; but the largest crystals of mica are to be found within 12 ft. of one schist wall and within 4 ft. of the other. This is not always the case; for at another of this company's mines the pegmatite is productive of large sheets of mica, usually extending from wall to wall. Comparatively few of the many pegmatite lenses in this district have crystallized coarsely enough to make the mica contained in them marketable. By far the greater number are of much finer texture. But there seems in this southern district to be no very marked gradual change from fine to coarser crystallization as the distance from the granite increases. The extremely coarse-grained pegmatites are dotted very sparsely among others of more normal texture; and, in fact, some are in close proximity to the granite itself.

A feature common to almost all the bodies of pegmatite, and also to the schist country in which they occur, is that the strike of the former and the cleavage of the latter are in most cases roughly at right angles to a line drawn from the center of the granite intrusion of Harney Peak. This remark is not intended to imply that the strike of the cleavage of the schists and that of the pegmatites, greisen, and quartz ledges is in every case coincident. There is sometimes a difference of from 10° to 40° between the two. Moreover, the schists almost always dip at a high angle away from the granite, while the ledges may dip either one way or the other; and in many cases where the dip of both ledge and schist is in the same direction, the angle is very different. The fact that the schists always tend to dip away from the granite and that their strike (except to the north of Hill City) is rudely parallel to the sides of the granite intrusion, indicates that the metamorphism of the schists has been to a great extent due to the intrusion of this mass of granite. But that the whole of the metamorphism of the district has been caused by this one factor is most improbable.

The ledges themselves furnish some evidence in this connection. I take as an example the ledge of massive greisen which has been developed in the Cowboy mine, near Hill City.

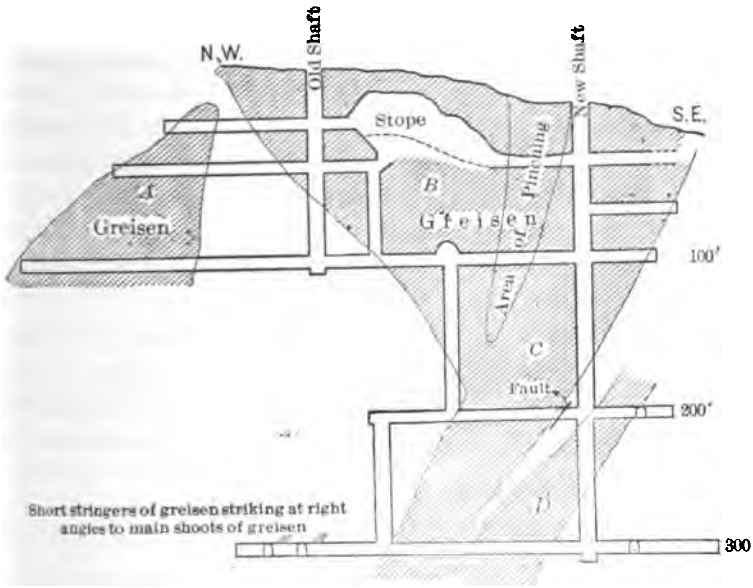


FIG. 3.—SECTION OF COWBOY MINE.

The arrangement of the mica and quartz is shown in Fig. 1. There is a difference between the strike of the greisen and that of the schist of about 30°. The former dips about 75° NE., while the latter dips about 65° NW. This might suggest a fissure-vein, were it not for the fact that the greisen has the appearance of being "frozen on" to the schist walls.

The longitudinal section of the mine, Fig. 3, shows the various shoots of greisen, and it will be seen how very lenticular they are in depth. The schist walls show numerous slickensides in many directions, and in some places slight contortions in dip and strike. A distinct fault is noticeable on the 200-ft. level, where the greisen of shoot *C* has been turned round at right angles near the shaft, and has "petered out" in the west wall, while on the 300-ft. level, at the far north end, two little stringers of greisen, with a strike at right angles (roughly) to that of the main shoots, have been followed by tunnels, which proved only a rapid "petering out" in each case. This clearly shows that a force acting at right angles to that which had produced the cleavage of the schists, had been applied to the rocks in the vicinity, after the deposition (if deposition we may call it at this stage) of the greisen.

A neighboring shoot of greisen, which strikes at an angle with the schists, shows the effects of this subsequent dynamic action, being crossed from wall to wall by cracks following the line of strike of the schists. Besides these two instances in which a second lateral disturbance is evidenced in the greisens, there are many others to be seen in the pegmatites also, although here we find the cracks in the ledges running rather at right angles to their strike than in the direction of the cleavage of the schists.

As I have already observed, the pegmatites showing greisen usually do so on the walls—*i. e.*, at the contact of the granitic rock with the schist. It is also not unusual to find the greisen in a crack or fissure in the pegmatite, in which case it is generally massive and rarely granular.

Possible Causes of Dynamic Action Subsequent to the Intrusion of the Main Granite Batholith.

As we have already seen, there is ample evidence to prove that, besides the intrusion of the Harney Peak granite, there were forces, acting in this region at a subsequent time and in other directions, which cracked and fissured the ledges of quartz, pegmatite, and greisen.

What produced these forces?

North of Harney Peak, and within a mile or so of the granite itself, we find several outcropping bosses of pegmatite very much coarser in texture than any of the ledges hereinbefore described. Their appearance suggests also that, unlike those ledges (which have been proved by underground prospecting in several places to have no greater continuity in depth than on the surface), they may have considerable downward extension, and even a deep-seated origin. I refer to masses of the type of the fairly well-known Etta boss or knob, which was the first to receive attention during the tin-excitement in the late nineties. Here we have a pegmatite (giving evidence which points to an intrusive origin), the constituent minerals of which have crystallized out on an enormous scale.

Professor Blake, who reported on this property about 1883, described the arrangement of the minerals in the boss as showing a tendency to more or less concentric segregation. This is to a certain extent true, although the lines of demarcation are less hard and fast than Blake's report suggests. He describes the zones as follows: (1) a central core of quartz and feldspar; (2) greisen, carrying tin-stone; (3) crystals of spodumene, with greisen, carrying tin-stone; (4) a band or zone of biotite and muscovite mica.

Spodumene is found, in this region, only in bosses of pegmatite of extraordinary coarseness, like this one. Furthermore, this intrusion contains not less than 48 distinct minerals, many of which are absent from the more common and more normally crystalline pegmatite ledges. Of the minerals occurring here, the most important, as regards size of segregated masses, are quartz, albite, spodumene, and mica (the latter being usually commercially useless, since it occurs as muscovite and biotite in "fishbacked" plates). Granular greisen is also prevalent, and in some spots, especially near the center of the boss, exhibits a certain amount of cassiterite, while aggregates of small crystals of mica occur in association with it.

That such occurrences of pegmatite have no connection with the main intrusion of granite, there is scarcely sufficient evidence to prove, but it does seem possible that their intrusion was connected with the action which produced some of the later lateral disturbances and uplifts. They have no real parallel southwest and south of Harney Peak, where the very coarsely crystalline pegmatites productive of commercial mica outcrop at various distances from the granite. There are some marked differences between the two groups. The latter show relatively fewer minerals; often occur as lenses in the schist, like the more normal pegmatites; and the mica in many of them is the variety known as phlogopite, which is generally common to most of the ledges in the near vicinity, and is the mica of some of their tin-bearing granular greisen. There seems to be, therefore, a distinction between the mormal pegmatites occurring in the Harney Peak district all round the granite, and the highly segregated intrusions to the north; and the question suggests itself, in view of the evidence of subsequent dynamic action portrayed in the fissured normal pegmatites and greisens, whether these other intrusions of coarser granitic material outcropping to the north are not the cause, or products of the cause, of the later disturbances and uplifts thus shown to have taken place. In connection with this question, the schists and slates which constitute the country-rock of the district may be considered.

Newton, in his report on the geology of the district, seems to think that the present surface-exposure of schists and slates represents from east to west the original thickness of the pre-Cambrian series. He says, in speaking of the body of Archæan slates and schists:[1]

> "It has not, of course, its original position; for the same process of change which has produced its metamorphic structure has turned it bodily on edge, and either broken or eroded away its upward continuation; . . . and it is probable that the system presents the clays, shales, and sandstones from which it was produced by metamorphism in the same order in which they were originally deposited."

Apparently, Newton thought there was some connection between the cleavage and foliation of the schists and the original bedding of the detritus from which they were formed. Against this supposition we have the following important facts:

1. The general strike of the cleavage of the slates and schists is parallel to the main granite batholith.

2. The general dip of the slates and schists is away from the granite.

3. The dip of the schists is generally extremely uniform. Had the present structure represented bedding, this state of things would have been most improbable; the folds would have had to be very closely pressed, and, as Van Hise says,[2]

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[1] The Pre-Cambrian Rocks of the Black Hills, by C. R. Van Hise, *Bulletin of the Geological Society of America*, vol. i., pp. 203 to 244 (1889).

[2] *Idem.*

"the sides . . . would have had to be truncated in such a fortunate position as to cut none of the folds at their turning-points."

4. In only one instance has a true anticlinal fold of large size been noted in the schists. This is in the neighborhood of the very coarsely crystalline pegmatites in the Custer City district, and is probably due to a purely local disturbance.

We can therefore say that the available evidence all points to the fact that the present structures exhibited in the pre-Cambrian slates and schists is the product of metamorphism, and has no relation whatever to their bedding; and, furthermore, that the forces producing the metamorphic structure, which have acted at various times and in various directions, have completely obliterated all traces of the original bedding of the detritus. Many of the pegmatite occurrences in the schist exhibit a marked flow-structure in the arrangement of their minerals. This is probably due to forces which have acted on them subsequent to their deposition. None of the ledges of pegmatite, quartz, or greisen have any great continuity along their outcrops. In some cases, where shafts have been sunk on them to a considerable depth, this has been found to be true as to their downward extension also. Most of those upon which the greatest development-work has been done show somewhat irregular pinching and widening and faulting, although the schist itself may not exhibit any marked contortion in the immediate vicinity—the only trace of the course of a fault underground being, in such cases, a very narrow gouge or clayey stringer.

Origin of Lenses of Pegmatite and Quartz.

The study of the origin of the pegmatite, quartz, and greisen in this district raises the question of the probable causes of pegmatization. Van Hise, in his work,[3] states three causes for this phenomenon, all of which may be applicable to the Harney Peak district, namely: (1) true igneous injection; (2) aqueo-igneous action (intermediate between solution and fusion); and (3) water-cementation.

As to the first cause, we have, in close proximity to the main granite intrusion of Harney Peak, several very large dikes, of composition and texture extremely similar to that of the granite batholith, but exhibiting a distinct gneissic structure. It seems impossible not to imagine that these dikes are connected in depth with the main granite intrusion, and owe their origin to igneous injection.

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[3] A Treatise on Metamorphism, *Monograph XLVII.*, *U. S. Geological Survey*, pp. 720 to 728 (1905).

Further away from the main granite, the minerals of the pegmatites become more and more segregated, and there is a distinct gradation from finer to coarser texture (although with very little degree of regularity in the ratio of increase in distance). Some of these occurrences of pegmatite, usually those in which a small percentage of cassiterite has been found, have been prospected for that mineral by shafts, which, in several cases, have proved their lenticular character in depth as well as in length by "bottoming" them. To these it seems reasonable to ascribe an aqueo-igneous origin, through the injection, along fissures, in a state between fusion and solution, of material more acid in composition than the granite magma, which has crystallized very coarsely when deposited. Van Hise puts forward the theory that the coarseness of crystallization at a distance from the main granite intrusion is not necessarily due to any great slowness in cooling, but rather to the aid to crystallization afforded by the abundance of mineralizers present.[4] This theory certainly answers for the majority of the phenomena of pegmatite lenses in this immediate district. Still further west and north from Harney Peak, the lenses of pegmatite become more quartzose and the number of lenses of pure quartz increases. West of Hill City we have several lenses of massive greisen or quartz containing mica, both materials being arranged in layers parallel to the walls of schist. The origin of these seems to be accounted for more easily by water-cementation than by any other cause; they are very lenticular and irregular in their occurrence, both in strike and in depth.

It is evident that no hard and fast line can be drawn where the effects of any one cause of the formation of the lenses of quartz and pegmatite begin or end. Their appearance seems to present a gradual gradation from the granitic dikes presenting a gneissic structure, owing their origin to igneous injection, and which occur in close proximity to the main granite intrusion, to the lenses of pegmatite, and thence to those lenses composed almost, if not entirely, of quartz, which occur remote from the granite, and probably owe their origin to water-cementation.

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[4] *Ibid.*, p. 722.

[TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS.]

Progress in Roll-Crushing.

Discussion of the paper of C. Q. Payne, presented at the New York meeting, February, 1912, and printed in *Bulletin* No. 66, June, 1912, pp. 577 to 602.

C. Q. PAYNE, New York, N. Y. (communication to the Secretary*): I desire to comment briefly upon S. Arthur Krom's contribution to the discussion of my paper. In giving Stephen R. Krom the credit of introducing the belted high-speed crushing-roll into the United States, I may say that the difference in speed of the early geared Cornish rolls, which were sometimes driven as slowly as 4 or 5 rev. per min., and that of the belted rolls, which, when run at 100 rev. per min. and over, may be considered "high speed," as compared with the former, seemed to me sufficiently distinctive to be noted. The same distinction is made by Philip Argall in his interesting paper, Sampling and Dry Crushing in Colorado.[1] The narrow-tired roll, which was driven at an extremely high speed, was a later development, and which proved to have other disadvantages.

I regret to notice (pp. 598, 599) that Mr. Krom makes a misleading use of quotation-marks in his comments by inclosing between them statements which do not quote accurately from my paper. However, I think this must be due to oversight or haste on his part, for I notice also that he entirely disregards Mr. Edison's remarkable use of rolls in pulverizing cement-rock, when he says (p. 598) that his experience has taught him that the product of a roll is always granular, no matter how fine the crushing is.

He has again overlooked my statement on p. 591, when he says (p. 599) that the Frazee rolls cannot be applied to general mill-practice owing to the necessity of drying the ore. No such necessity exists. I must also notice that while Mr. Krom classes the Frazee rolls with Joplin practice among things to be condemned, he finds the correct roll-construction in the Krom roll. As nothing is stated about feeding the Krom roll, this conclusion seems to be too general to be of value in this connection, or quite convincing, even if it might not have been safely inferred.

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* Received June 25, 1912.

[1] *Transactions of the Institution of Mining and Metallurgy*, vol. x., p. 250 (1901–02).

In regard to Joplin practice, the use of flanges on one of the roll-shells in some of the mills is certainly an interesting development. No feed-mechanism is, however, employed. The ore is carried to the rolls by a stream of water, and the object of the flanges is to direct the ore downward, and to prevent the water from splashing it out sideways while it is being crushed. These rolls are not used for fine crushing in the sense that Mr. Frazee's rolls are, and the flanges do not prevent grooving and irregular wear of the shells. The latter are made of cast-iron, and are given a very hard chill. When the chill is worn through they are discarded. As Mr. Stone points out, the distinctively novel feature of the Frazee roll-design is the method of feeding the rolls. Here the feed-stream is extended slightly beyond the ends of the shorter roll, so that by a certain amount of crushing between the ends of the roll and the flanges, the ridging of the ends of the shorter roll, as well as grooving, can be entirely overcome. The Frazee roll-shells can thus be brought quite close together, and in this way used for much finer crushing than the Joplin practice would permit. They have, therefore, little or nothing in common with the latter.

I am aware that it is not unusual for manufacturers of machinery to take a fling at Joplin practice. From a mining engineer's point of view, however, it has much to commend it. The Missouri zinc-deposits, for the most part, are irregular and uncertain in extent. Much of the ground has heretofore been leased to operators in comparatively small sections. The expense of simple mill-equipments can be more quickly recovered from operating-profits than more elaborate and expensive machinery. It is always "good practice" to make money, and rolls which would not be advisable for porphyry copper-ores, for example, under the peculiar conditions surrounding zinc-mining at Joplin, have held their own, because they have justified themselves financially.

In my paper I have endeavored to press the point, that in treating those ores which must be kept in a granular condition while crushing them finely for concentration, the Frazee rolls hold out the promise of great usefulness by avoiding losses due to sliming. In Mr. Krom's comment I notice that he gives the screen-analysis of a material which has passed through a Hardinge tube-mill, but which does not have to be kept in a granular condition for concentration. The reference, therefore, does not seem pertinent to anything contained in my paper.

In regard to Mr. Krom's opinion on the subject of coarse crushing, his comparison between rolls and jaw-crushers on the basis of assumed cost seems decidedly unfair to the rolls.

I regret that Mr. Krom should rely for his opinion of Mr. Edison's giant rolls upon a correspondent who seems to have been unfortunate in not having secured the benefit of their enormous capacity, and who does not state the nature of the rock crushed. So indirect an attack may well be disregarded. Mr. Edison's mechanical genius does not need any tribute at my hands. His is the faith (in mechanics) which literally removes mountains, and no one can watch the giant rolls in operation at Stewartsville, N. J., without increased respect for the human spirit which can measure itself with the enormous forces which the rolls unlock.

Whether the particular design developed thus far is found to be the best for the purpose or not, there is no doubt that all mining engineers will watch with keen interest the reduction of costs in mining and quarrying by means of coarse crushing, in which Mr. Edison has led the advance. He has certainly made a most notable contribution to the art of roll-crushing.

[TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS.]

The Refining of Blister-Copper.

Discussion of the paper of Horace H. Emrich, presented at the New York meeting, February, 1912, and printed in *Bulletin* No. 66, June, 1912, pp. 611 to 632.

ANTON EILERS, New York, N. Y. (communication to the Secretary*):—I do not think there is anything to be added to this paper at the present time. The facts as given are no doubt correct, for I know from personal contact that Mr. Emrich was a close observer and a logical thinker.

But this paper carries a suggestion to those who work in this particular line. The subject is by no means exhausted. It should be the ambition of some investigator to find out definitely, why, when slimes are caught as such in the copper during its deposition, the conductivity of the copper is not impaired, if the tests are made directly from the cathode, while it is lowered to a marked degree after the same cathode-copper has been melted down, even if an attempt to refine it further by oxidizing-heat has been made.

To me this behavior of the cathode-copper before and after melting indicates that the impurities in the slimes are present in a different state before and after melting down, and that the arsenic and possibly the antimony present play an important part. I would therefore suggest that a number of complete analyses of the slimes afloat in the first overflow from the tanks should be carefully made, particularly to find out in what combination, if any, arsenic and antimony are present; that another number of complete analyses of the melted cathode-copper be made for the same purpose, and that these analyses be supplemented by metallographic tests of respectively the unmelted and the melted cathode-copper. The results would probably indicate steps to be taken to reduce the difference in conductivity to a considerable degree. Without the above and other tests resulting from these, we are at present in the dark. Yet it is clear that large commercial advantages may be gained if the above questions are favorably settled.

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* Received Mar. 22, 1912.

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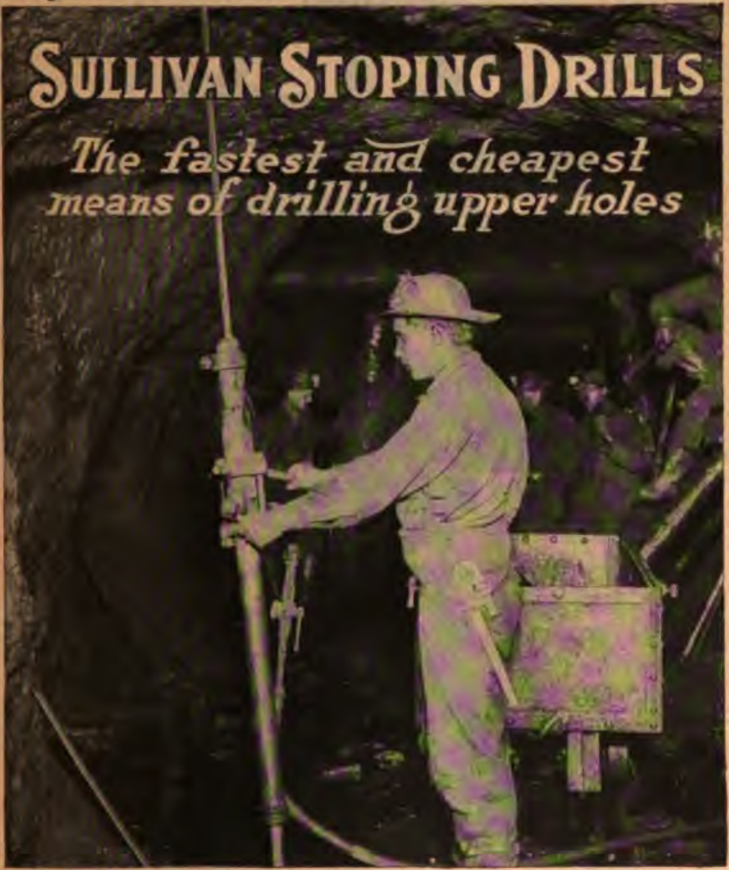
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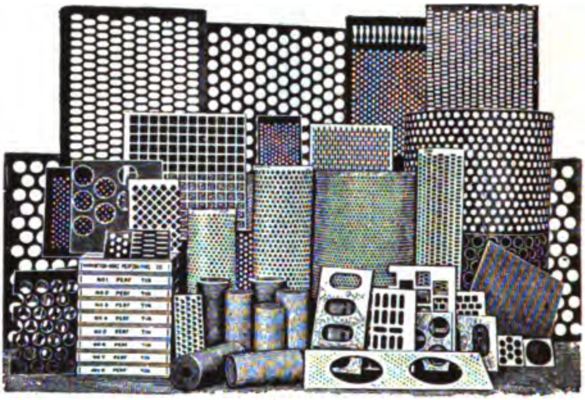

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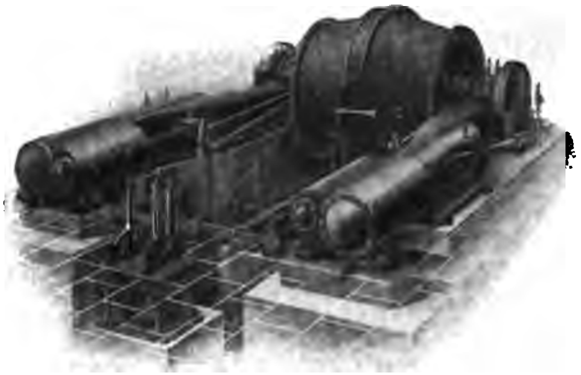

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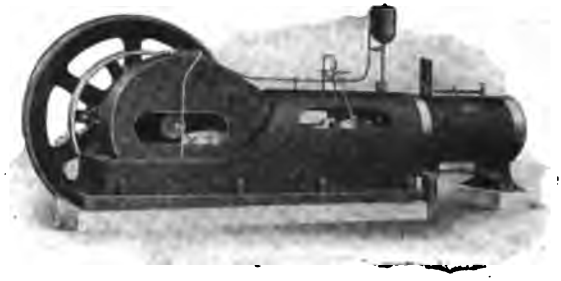

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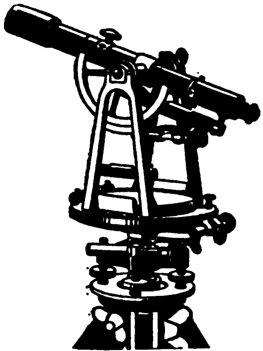

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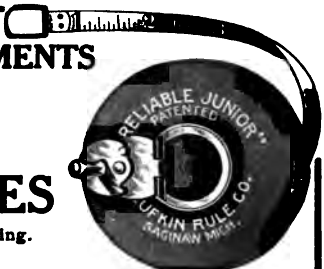

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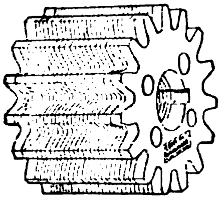

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No. 68 AUGUST 1912

Bulletin of the American Institute of Mining Engineers.

CONTENTS:

SECTION I.—INSTITUTE ANNOUNCEMENTS.

PAGE
List of Officers and Committees, ... ii
Institute Announcements, ... v
Library, ... xii
Membership, ... xix

SECTION II.—PAPERS AND DISCUSSIONS.

The Treatment of Complex Silver-Ore at the Lucky-Tiger Mine, El Tigre, Sonora, Mexico. By D. L. H. Forbes, ... 769
Alaska Coal-Land Problems. By H. Foster Bain. With Discussion by R. W. Raymond and William H. Hampton, ... 811
The San Nicolás Mining District, San Nicolás, Tamaulipas, Mexico. By Irving H. Wentworth, ... 843
Gold-Mines in Southern Colombia. By Fortunato Pereira Gamba, . 853
The Occurrence of Silver-, Copper-, and Lead-Ores at the Veta Rica Mine, Sierra Mojada, Coahuila, Mexico. By Frank R. Van Horn, . 867
Vanadium in Pig-Iron. By Porter W. Shimer, ... 883
Discussion of Mr. Young's paper, Slime-Filtration. By Askin M. Nicholas, ... 887

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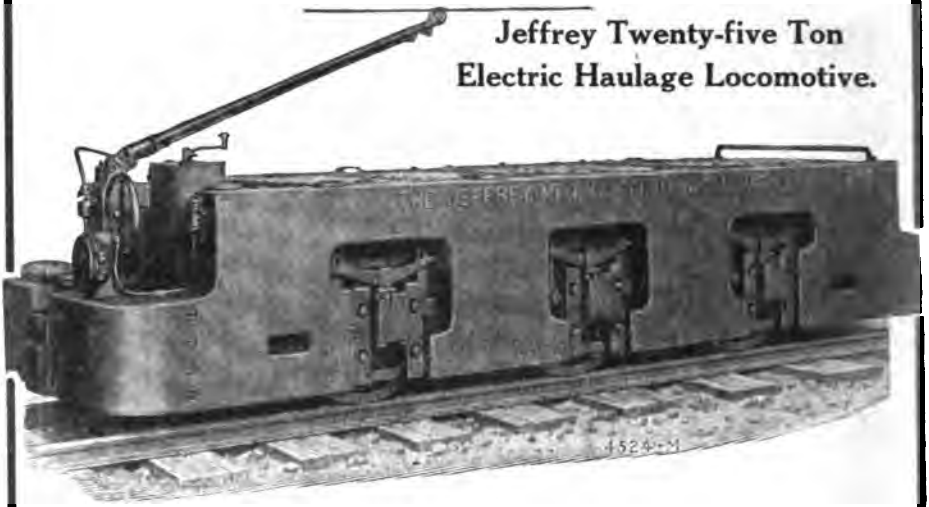

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TABLE OF CONTENTS.

SECTION I. INSTITUTE ANNOUNCEMENTS.

PAGE
List of Officers and Committees for the Year Ending February, 1913, . ii
Institute Announcements, ... v
Library, ... xii
Membership, ... xix

SECTION II. TECHNICAL PAPERS.

[The American Institute of Mining Engineers does not assume responsibility for any statement of fact or opinion advanced in its papers or discussions.]

No. 1. The Treatment of Complex Silver-Ore at the Lucky-Tiger Mine, El Tigre, Sonora, Mexico. By D. L. H. FORBES, ... 769
No. 2. Alaska Coal-Land Problems. By H. FOSTER BAIN. With Discussion by R. W. Raymond and William H. Hampton, ... 811
No. 3. The San Nicolás Mining District, San Nicolás, Tamaulipas, Mexico. By IRVING H. WENTWORTH, ... 843
No. 4. Gold-Mines in Southern Colombia. By FORTUNATO PEREIRA GAMBA, ... 853
No. 5. The Occurrence of Silver-, Copper-, and Lead-Ores at the Veta Rica Mine, Sierra Mojada, Coahuila, Mexico. By FRANK R. VAN HORN, ... 867
No. 6. Vanadium in Pig-Iron. By PORTER W. SHIMER, ... 883
No. 7. Discussion of Mr. Young's paper, Slime-Filtration. By ASKIN M. NICHOLAS, ... 887

All communications concerning the contents of this Bulletin should be addressed to JOSEPH STRUTHERS, Ph.D., Secretary and Editor, 29 W. 39th St., New York, N. Y.

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Eighth International Congress of Applied Chemistry, New York, September, 1912:—William L. Saunders, George C. Stone.

Sixth Congress of the International Association for Testing Materials, September, 1912:—Robert Forsyth.

Committee No. 24, International Association for Testing Materials:—Henry D. Hibbard.

International Engineering Congress, San Francisco, 1915:—Samuel B. Christy, William C. Ralston, Edwin T. Blake.

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Conference to Discuss Patent Laws:—A. F. Lucas, ————, ————.

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Any society of undergraduates at a technical school, comprising students in any branch of engineering, metallurgy, chemistry, geology, etc., may be recognized by the Council in its discretion as an Affiliated Student Society. A circular giving details of the plan of affiliation may be obtained on application to the office of the Secretary of the Institute.

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INSTITUTE ANNOUNCEMENTS.

Special Notice.

The *Bulletin* is now entered at the Post Office at Second-Class Postage rate of one cent per pound, and in order to preserve this privilege it is necessary that the dues of members be paid within four months of January 1 of each year. If the dues are not paid within the period mentioned, a member's name must be removed from the regular subscription-list and the *Bulletin* mailed at the transient second-class postage rate of one cent for each four ounces or fraction thereof, prepaid by stamps affixed. It is therefore earnestly requested that dues be paid promptly—otherwise the Institute will be put to additional expense of postage and to added labor in removing and replacing names from the regular list, and in maintaining an additional separate mailing-list.

Adjourned Annual Business Meeting of the Institute.

At the Annual Business Meeting of the Institute, Feb. 20, 1912, it was voted to postpone the consideration of the proposed amendments to the Constitution until June 3, 1912, and the meeting adjourned to reconvene on that date, or on such subsequent date as should be fixed by the Special Committee of Five appointed on Feb. 20, and the Board of Directors of the Institute, acting jointly.

Due to various causes, including absence of two members of the Committee in Europe for two months, it has not been possible for the Special Committee of Five to complete its report in time for preliminary distribution to members of the Institute, and consideration by them prior to June 3, 1912, and it has been mutually agreed by the Special Committee and the Board of Directors to postpone the date at which the adjourned meeting shall be reconvened from June 3, 1912, to Oct. 7, 1912.

Nominating Committee.

At the meeting of the Council, June 28, 1912, the President announced the appointment of the following Nominating Committee to present names for officers of the Institute to be voted upon at the Annual Business Meeting, Feb. 18, 1913.

Philip N. Moore, St. Louis, Mo., *Chairman.*
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Joseph W. Richards,
Albert Sauveur,
F. W. C. Schniewind,
A. A. Stevenson,
Felix A. Vogel,
Leonard Waldo,
William R. Webster.

The following revised list of iron and steel papers will, according to present indications, be presented at the coming meeting of the Institute, to be held in Cleveland, Ohio, the latter part of October.

Some of these have already been received and are being printed. They will be sent at an early date to all members known to be interested in iron and steel, and will be sent to any member of the Institute applying for them. The same procedure will be adopted with other papers received, and it is hoped to send all out well in advance of the meeting, so as to secure a good discussion, the present indications being that a number of prominent iron and steel men will attend the meeting.

Notes on Ruff's Carbon-Iron Equilibrium Diagram, by Henry M. Howe, New York, N. Y.

The Case-Hardening of Special Steels, by Albert Sauveur and G. A. Reinhardt, Cambridge, Mass.

The Heat-Treatment of Steel, papers by Henry M. Howe, New York, N. Y.; R. R. Abbott, and Mark Ammon, Cleveland, Ohio.

The Effect of Alumina in Blast-Furnace Slags, by J. E. Johnson, Jr., Ashland, Wis.

Blowing-In of Blast-Furnaces, by Ralph H. Sweetser, Columbus, Ohio.

On Blast-Furnace Air-Pressures, by J. E. Johnson, Jr., Ashland, Wis.

The Utility of Efficiency-Records in the Manufacture of Iron, by John J. Porter, Staunton, Va.

A Novelty in Open-Hearth Steel-Furnace Practice, by N. E. Maccallum, Phœnixville, Pa.

Notes on Titanium, and on the Cleansing-Effect of Titanium on Cast-Iron, by Bradley Stoughton, New York, N. Y.

Cuyuna Iron-Ore Range, by Walter A. Barrows, Jr., Duluth, Minn.; and Carl Zapffe, Brainerd, Minn.

Method of Surveying and Sampling Diamond-Drill Holes, by E. E. White, Glen White, W. Va.

The Concentration of Iron-Ores, by N. V. Hansell, New York, N. Y.

Iron-Ore, by Dwight E. Woodbridge, Duluth, Minn.

The Concentrating- and Briquetting-Plant of the Moose Mountain, Limited, by Charles E. Hermann, New York, N. Y.

The Use of Schumacher Briquettes in the Johnstown Blast-Furnaces, by Joseph W. Richards, So. Bethlehem, Pa.

Rational Cupola-Practice, by Richard Moldenke, Watchung, N. J.

The Manufacture of Coke, papers by F. E. Lucas, Sydney, C. B., Canada; W. H. Blauvelt, Syracuse, N. Y.; and C. W. Andrews., Duluth, Minn.

On Mayari Steel.

The Heat-Treatment of Steel-Castings, by C. D. Young, Rolla, Mo.

Notes on the Structure of Steel, by William Campbell, New York, N. Y.

A printed list of members interested in iron and steel has been sent out, with requests for any notes for either change or correction thereof.

The Committee has been much gratified by the many answers received to its recent circular letter sending out these lists and asking for the co-operation of the members of the Institute interested in, or concerned with, iron and steel.

Reception to the International Association for Testing Materials.

The Sixth Congress of the International Association for Testing Materials will be held in the Engineering Societies Building, 25–33 West 39th Street, Sept. 2 to 7.

On the evening of the opening of the Congress, Monday, Sept. 2, from nine to ten, a reception will be tendered to the Congress by the American Society for Testing Materials, the three National Societies housed in the building, viz., the American Institute of Electrical Engineers, the American Society of Mechanical Engineers, and the American Institute of Mining Engineers, co-operating.

The reception is in charge of the following Committee:

For the American Society for Testing Materials:

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MAJOR WM. H. WILEY,
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For the American Institute of Mining Engineers:

PROF. JAMES F. KEMP, *President.*

MR. CHARLES KIRCHHOFF,
PROF. JOSEPH W. RICHARDS,
MR. E. GYBBON SPILSBURY,
DR. LEONARD WALDO.

The reception will be formal, evening dress being expected.

On this occasion the rooms of the various Societies on the different floors, together with the library, auditorium, etc., will be thrown open for the inspection of the guests.

In addition to the general reception on the first floor, there will be a welcome extended to the visitors by each Society in its own rooms.

All members of the Institute and ladies are cordially invited to attend.

Eighth International Congress of Applied Chemistry.

The Eighth International Congress of Applied Chemistry will be held at Washington, D. C., and New York, Sept. 4 to 13, 1912. Joint sessions of the American Institute of Mining Engineers and Sections III, IIIa, of the Congress are to be held at Columbia University, New York, Sept. 6 and 7. The papers for the joint sessions will be presented according to the following provisional program.

Contributed by the Congress.

FRIDAY, Sept. 6.

10:00 A.M.

Section III, IIIa, Mining and Metallurgy.

R. Adan: Micrographie des Fontes Speciales de Fer.

J. A. Matthews: Some Physical Characteristics of High-Speed Tool-Steel.

E. B. Cook: Further Experience with the Gayley Dry-Air Blast.

R. B. Carnahan: Open-Hearth Steel-Practice.

A. Raman: Improvements on Kilns for Briquetting Iron-Ores.

A. Weiskopf: (1) Ueber den gegenwaertigen Stand der Eisenerzbrikettierung. (2) Einfluss von Eisenerz briketts auf den wirthschuflichen Betrieb des Hochofens.

A. S. Dwight: The Dwight & Lloyd Process of Roasting and Sintering.

A. Lidoff: Iron Coke.

1:00 P.M.

Section III, IIIa, Mining and Metallurgy.

E. P. Mathewson: Development of the Reverberatory Furnace for Smelting Copper-Ores.

F. Laist: Chemistry of the Reduction-Processes in Use at Anaconda, Mont.

Utley Wedge: Roasting Copper-Ores and Concentrates Preliminary to Leaching, to Secure Copper-Values.

W. M. Corse: Modern Manganese-Bronze Tests.

Robert C. Canby: Development of the American Water-Jacket Lead Blast-Furnace.

F. Cazin: The Evolution of the Modern Lead Blast-Furnace.

F. Guiterman: Silver-Lead Smelting.

E. F. Eurich: Zinc-Desilverization.
——— Skinzopoulos: Un Nuovo Processo di Panificazione di Sterri e Sabbie di zolfo a ganga priva di calce ed Argille.

SATURDAY, Sept. 7.

10:00 A.M.

(Section Xa, Electro-Chemistry.)

F. A. J. FitzGerald: Heat-Losses in Furnaces.
C. H. Von Bauer: Electrolytic Induction and Resistance Furnaces.
A. E. Greene: Electric Furnaces in the Metallurgy of Steel.
P. Heroult: Recent Developments of the Electric Steel-Furnace.
A. E. Greene: Electric Heating and the Removal of Phosphorus from Iron.
R. Amberg: The Function of Slag in Electric Steel-Refining.
F. T. Snyder: The Electric Smelting of Zinc-Ore.
J. O. Handy: Extraction of Copper from Sulphide Ores by Roasting, Leaching, and Deposition.
L. Addicks: Cathode-Impurities in Copper-Refining and Their Sources.

Contributed by the Institute.

(Time and place not yet assigned.)

Henry M. Howe, New York, N. Y.: Notes on Ruff's Carbon-Iron Equilibrium Diagram.
N. S. Maccullum, Phœnixville, Pa.: A Novelty in Open-Hearth Steel-Furnace Practice.
Ralph H. Sweetser, Columbus, Ohio: Blowing-In of Blast-Furnaces.
F. E. Lucas, Sydney, C. B., Can.; W. H. Blauvelt, Syracuse, N. Y.; C. W. Andrews, Duluth, Minn.: Manufacture of Coke.
Bradley Stoughton, New York, N. Y.: Notes on Titanium, and on the Cleansing-Effect of Titanium on Cast-Iron.
Walter A. Barrows, Jr., Duluth, Minn.; and Carl Zapffe, Brainerd, Minn.: Cuyuna Iron-Ore Range.
Albert Sauveur and G. A. Reinhardt, Cambridge, Mass.: Case-Hardening of Special Steels.
William Campbell, New York, N. Y.: Some Metallographic Notes.
H. O. Hofman and W. Wanjukow, Boston, Mass.: Decomposition of Metallic Sulphates at Elevated Temperatures in a Current of Dry Air.
J. E. Johnson, Jr., Ashland, Wis.: The Effect of Alumina in Blast-Furnace Slags.

A New Volume on Ore-Deposits.

Ore-Deposits—a continuation of the "Posepny" Volume.

Comprising Papers Descriptive of Ore-Deposits and Discussions of heir Origin, Edited, with an Introduction, by Dr. S. F. Emmons. The volume contains also a Biographical Notice of Dr. Emmons by his associate and friend, Dr. George F. Becker, and a comprehensive Bibliographical Index of the Science of Ore-Deposits, prepared by

Prof. John D. Irving, of the Sheffield Scientific School of Yale University. Dr. Emmons had finished his editorial work and written his Introduction before his lamented death in 1910; and the Volume contains his last words upon the subject to which he had given the work of his life, and on which he was justly regarded as the foremost authority.

Further details concerning time of publication, price, etc., will follow.

Library Research-Work.

The attention of members of the Institute is again directed to the research-work done by the librarian and his assistants, which should attract special attention from those members who have no access to the literature of subjects in which they may be interested.

During the year 1911 there were 143 searches made for members and non-members of the Founder Societies, and copies of the references have been preserved for the use of others. This work has been largely based on requests sent in by mail, from Japan, South Africa, Mexico, Canada, and England, as well as from different parts of the United States. The Librarian is confident that if it were more widely known that the library is equipped to undertake researches, the demand would increase beyond the ability of the present force to handle it. The library receives more than 700 technical periodicals which are available through the indexes for this special purpose.

Back Volumes of the Transactions.

The Board of Directors has authorized the following offers of sets of back volumes of the *Transactions*, at considerably reduced prices, to Members, Libraries, and Scientific Societies:

		Per Set.
I.	Five volumes, bound in half-morocco, from No. 36 (1906) to No. 40 (1910), ...	$20
II.	Ten volumes, bound in half-morocco, from No. 31 (1902) to No. 40 (1910), including Mexican Volume, . .	35
III.	Twenty volumes, bound in half-morocco, from No. 21 (1893) to No. 40 (1910), ...	50
IV.	Thirty volumes, bound in half-morocco, from No. 11 (1883) to No. 40 (1910), ...	60
V.	Thirty-nine volumes, bound in half-morocco, from No. 1 (1873) to No. 40 (1910), with the exception of No. 10 (1882), but including index for Volumes Nos. 1 to 35, and Nos. 36 to 40, ...	75
VI.	Nine volumes, bound in half-morocco, from No. 1 (1873) to No. 9 (1881), ...	25

Applications should be addressed to Joseph Struthers, Secretary, 29 West 39th Street, New York, N. Y.

The Emmons Research Fellowship of Economic Geology.

The Committee named below has been formed by friends of Samuel Franklin Emmons, late of the United States Geological

Survey, to consider the best method of perpetuating his name. It has been decided that the memorial to him shall take the shape of a Research Fellowship, to be known as the Samuel Franklin Emmons Research Fellowship of Economic Geology, which is to be administered by Prof. James F. Kemp, of Columbia University, New York. Subscriptions are invited by his friends to this fund, which the Committee has fixed at $25,000.

Members of the Institute who desire to contribute to this fund will please communicate with the Treasurer, Benjamin B. Lawrence, 60 Wall Street, New York.

The Committee consists of the following:

GEORGE OTIS SMITH, Director, U. S. Geological Survey, Washington, D. C.
H. L. SMYTH, Harvard University, Cambridge, Mass.
JAMES DOUGLAS, 99 John Street, New York, N. Y.
J. A. HOLMES, Director, Bureau of Mines, Washington, D. C.
JAMES F. KEMP, Columbia University, New York, N. Y.
F. W. BRADLEY, San Francisco, Cal.
J. PARKE CHANNING, 42 Broadway, New York, N. Y.
SEELEY W. MUDD, 1001 Central Building, Los Angeles, Cal.
D. W. BRUNTON, Denver, Colo.
H. FOSTER BAIN, 420 Market Street, San Francisco, Cal.
T. A. RICKARD, London, England.
B. B. LAWRENCE, 60 Wall Street, New York, N. Y.

How to Use the "Transactions" of the Institute.

Buy a copy of the Complete Analytical and Alphabetical Index of Volumes I. to XXXV., inclusive; also the new Index of Volumes XXXVI. to XL.

Whether you do or do not own a full set of the *Transactions*, these Indexes will make all of the material contained in the forty volumes available at once without detailed research into each volume separately. Moreover, an easy search will show what particular papers you need to know more about, and perhaps to study. Thus, any person possessing these Indexes can ascertain at once what has been published in the *Transactions* on a given question, and can learn, by writing to the Secretary, what is its nature, whether it is still to be had in pamphlet form, where it can be consulted in a public library, at what cost it can be copied by hand, etc., etc.

In short, to those who own complete sets of the *Transactions*, these Indexes will be a great convenience; but to those who do not, they will be a professional necessity.

The Index Volumes I. to XXXV. is an octavo of 706 pages, containing more than 60,000 entries, duly classified with sub-headings, and including abundant cross-references. The limited edition is becoming exhausted. The new Index, Volumes XXXVI. to XL., supplementing the Index Volumes I. to XXXV., brings the classified references up to the date of Volume XL., June, 1910. Prices: Index Volumes I. to XXXV., bound in cloth, $5; bound in half-morocco, to match the *Transactions*, $6. Index Volumes XXXVI. to XL., bound in cloth, $1.50; bound in half-morocco, $2.50. The delivery charges will be paid by the Institute on receipt of the above price.

LIBRARY.

AMERICAN INSTITUTE OF ELECTRICAL ENGINEERS.
AMERICAN SOCIETY OF MECHANICAL ENGINEERS.
AMERICAN INSTITUTE OF MINING ENGINEERS.
UNITED ENGINEERING SOCIETY.

WILLIAM P. CUTTER, Librarian.

The libraries of the above-named Societies are open from 9 A.M. to 9 P.M. on all week-days, except holidays, from September 1 to June 30, and from 9 A.M. to 6 P.M. during July and August.

The Library contains about 50,000 volumes, including sets of technical periodicals and the publications of scientific and technical societies.

The members of the Institute, with few exceptions, are by the very nature of their profession forced to spend a large portion of their time in localities isolated from sources of information. To such members the Library can render valuable service through correspondence, and letters requesting information will receive special attention. The Library is prepared to furnish references and copies of articles on mining and metallurgical subjects; to determine, if possible, the existence of mining-maps, and to furnish general information as to the geology and mineral resources of all countries as far as these resources are known and published.

It is hoped that the members of the Institute will avail themselves freely of this special service. The Library will welcome inquiries on engineering subjects, and furnish information as far as such information is to be obtained.

All communications should be made as definite as possible so that the information received may be what is desired and not include collateral matter which may not be of interest. In this way the time spent in searching for such collateral matter will be saved, and as a result the information will be sent more promptly and in more usable shape.

The members of the Institute can be of service to the Library by forwarding copies of mining-reports, maps privately issued, and similar material, which will be classified, indexed, and made available to other members.

Suggestions for additions to the Library, either by purchase or personal solicitation as gifts, will be welcomed. It is hoped that members while in the city will use the Library freely, and assurance is given that most careful service will be rendered to them.

Library Accessions.

July 1 to July 3, 1912.

[Copies of the list of additions to the Libraries of the American Society of Mechanical Engineers and the American Institute of Electrical Engineers can be obtained on application to the Secretary of the American Institute of Mining Engineers.]

Adressbuch 1912 Sämtlicher Bergwerke, Hütten- und Walzwerke Deutschlands nebst der Hebenbetriebe. VIII. Jahrgang. Leipzig, n. d. (Purchase.)

Advanced Cyanide Practice and Equipment. (Catalogue No. 10–C.) Denver, n. d. (Gift of Colorado Iron Works Co.)

Air Compressors and Blowing Engines Especially Adapted for Engineers. By C. H. Innes. London, 1906. (Purchase.)

Aitken's Manual of the Telephone. By W. Aitken. London, 1911. (Purchase.)

Allison. A Paper prepared and read before the Historical Society of Schuylkill County, Pa. By Robert Allison. April 24, 1912. N. p., n. d. (Gift of Author.)

American Iron and Steel Association. Annual Statistical Report for 1911. Part I. Philadelphia, 1912. (Exchange.)

Annuario Biografico del Circolo Matematico di Palermo, 1912. Palermo, 1912. (Gift of Rendiconti del Circolo Matematico di Palermo.)

Die Bergwerksmaschinen. Band II. By H. Bansen. Berlin, 1912. (Purchase.)

Bibliographie der Deutschen Zeitschriften Literatur. Band XXVIIIA. (Erganzungs-Band III.) Leipzig, 1912. (Purchase.)

Die Binären Metallegierungen. Teil I. By K. Bornemann. Halle a/S., 1909. (Purchase.)

Caisson Sickness and the Physiology of Work in Compressed Air. By Leonard Hill. London, 1912. (Purchase.)

Cast Iron in the Light of Recent Research. By W. H. Hatfield. London, 1912. (Purchase.)

Chemical Works, Their Design, Erection, and Equipment. By S. S. Dyson and S. S. Clarkson. London, 1912. (Purchase.)

The Coal Mines Act, 1911, and Other Acts Affecting Mines and Quarries. With a Commentary. By R. F. MacSwinney and P. Lloyd-Greame. London, 1912. (Purchase.)

Congreso Cientifico (1° Pan Americano) Agronomia y Zootechia. Tomo II. Santiago de Chile, 1912. (Gift of Congreso Cientifico.)

Crystallography and Practical Crystal Measurement. By A. E. H. Tutton. London, 1911. (Purchase.)

Diamond Drilling at Point Mamainse, Province of Ontario. (Bulletin No. 6, Canada Mines Department.) Ottawa, 1911. (Exchange.)

Einführung in die Technische Elektrochemie. Band I. By Paul Askenasy. Braunschweig, 1910. (Purchase.)

Emery and the Emery Industry. By A. Haenig, translated from the German by Chas. Salter. London, 1912. (Purchase.)

Engineering Association of New South Wales. Minutes of Proceedings. Vol. XXV. Sydney, 1912. (Exchange.)

Expériences sur le Travail des Machines-Outils. Alésage. By C. Codron. (Extrait de la Revue de Mécanique (1906–10.) Paris, 1912. (Purchase.)

Les Explosifs Modernes. Ed. 3. By P. F. Chalon. Paris, 1911. (Purchase.)

Fabrication et Emploi des Matériaux et Produits Réfractaires utilisés dans l'industrie. By Albert Granger. Paris, 1910. (Purchase.)

Fuel and Refractory Materials. By A. H. Sexton. London, 1909. (Purchase.)

GAZETTEER OF SURFACE WATERS OF CALIFORNIA. Part I.—Sacramento River Basin. (Water Supply Paper No. 295, U. S. Geological Survey.) Washington, 1912. (Exchange.)

GEMEINFASSLICHE DARSTELLUNG DES EISENHÜTTENWESENS. Herausgegeben vom Verein Deutscher Eisenhüttenleute in Düsseldorf. 8 Auflage. Düsseldorf, 1912. (Purchase.)

GENERAL FOUNDRY PRACTICE. By William Roxburgh. London, 1910. (Purchase.)

GEOLOGY AND ORE-DEPOSITS OF THE PARK CITY DISTRICT, UTAH. (Professional Paper No. 77, U. S. Geological Survey.) By J. M. Boutwell. Washington, 1912. (Exchange.)

GEOLOGY OF THE DUN MOUNTAIN SUBDIVISION, MELSON. (Bulletin No. 12, New Zealand Geological Survey Branch.) Wellington, 1911. (Exchange.)

HANDBUCH DER VERMESSUNGSKUNDE. By W. Jordan, fortgesetzt von C. Reinhertz. Erster Band. Stuttgart, 1910. (Purchase.)

HEADWATER REGIONS OF GULKANA AND SUSITNA RIVERS, ALASKA. (Bulletin No. 498, U. S. Geological Survey.) Washington, 1912. (Exchange.)

HYGIÈNE DE L'INDUSTRIE DU FER, MINES, HAUTS-FOURNEAUX, ACIÉRIES, FONDERIES, ETC. By Robert André. Paris, 1909. (Purchase.)

INDUSTRIAL PHILADELPHIA, FROM THE INFANT INDUSTRIES OF TWO CENTURIES AGO TO THE GIANT OF TO-DAY. By J. J. Macfarlane. Philadelphia, 1912. (Exchange.)

INTRODUCTION TO A HISTORY OF IRONMAKING AND COAL-MINING IN PENNSYLVANIA. By J. M. Swank. Philadelphia, 1878. (Gift of Author.)

IOWA STATE COLLEGE. Directory of Graduates of the Division of Engineering. Ames, 1912. (Gift of Iowa State College.)

JAARBOEK VAN HET MIJNWEZEN IN NEDERLANDSCH OOST INDIË, 1910. Batavia, 1912. (Exchange.)

DER KONSTRUKTIONSSTAHL UND SEINE MIKROSTRUKTUR. By A. Haenig. Berlin, 1910. (Purchase.)

KURZE UEBERSICHT UBER SÄMTLICHE LEGIERUNGEN. By Ernst Jänecke. Hannover, 1910. (Purchase.)

DIE LAGERSTÄTTEN DER NUTZBAREN MINERALIEN UND GESTEINE NACH FORM, INHALT UND ENTSTEHUNG. I. Band. Stuttgart, 1910. (Purchase.)

LAKE SUPERIOR IRON-ORE SHIPMENTS FOR 1911 AND PRECEDING YEARS BY MINES AND RANGES. Cleveland, 1912. (Purchase.)

LECTURES ON IRON FOUNDING. By Thomas Turner. Ed. 2. London, 1911. (Purchase.)

DIE LEGIERUNGEN IHRE HERSTELLUNG UND VERWENDUNG FÜR GEWERBLICHE ZWECKE. By G. Fermum. Hannover, n. d. (Purchase.)

LA MÉTALLOGÉNIE DE L'ITALIE ET DES REGIONS AVOISINANTES. By L. de Launay. Mexico, 1906. (Purchase.)

METALLOGRAPHY. By C. H. Desch. London-New-York, 1910. (Purchase.)

METALLURGY. By Ad. Fenchel, translated by H. J. Morris. London, 1911. (Purchase.)

METALLURGY OF STEEL. Ed. 4. By F. W. Harbord and J. W. Hall. Vols. 1–2. London, 1911. (Purchase.)

MICA, ITS OCCURRENCE, EXPLOITATION, AND USES. Ed. 2. Ottawa, 1912. (Exchange.)

MILLING MACHINES AND MILLING PRACTICE. A Practical Manual for the Use of Manufacturers, Engineering Students, and Practical Men. By D. De Vries. London, 1910. (Purchase.)

MINERAL INDUSTRY OF NEW JERSEY FOR 1911. (Bulletin No. 7, New Jersey Geological Survey.) Trenton, 1912. (Exchange.)

MINING AND METALLURGICAL SOCIETY OF AMERICA. Constitution, By-Laws, Rules, Officers and Members, Jan. 1, 1912. (Gift of Mining and Metallurgical Society of America.)

MODERN COKING PRACTICE, INCLUDING THE ANALYSIS OF MATERIALS AND PRODUCTS. By T. H. Byrom and J. E. Christopher. London, 1910. (Purchase).

MODERN IRON FOUNDRY PRACTICE. By Geo. R. Bale. Pts. 1–2. London, 1912, 1905. (Purchase.)

NEUERE ROHÖLMOTOREN. Teil I. By Ch. Pöhlmann. Berlin, 1912. (Purchase.)

NEW JERSEY. GEOLOGICAL SURVEY. Bulletin No. 7. Trenton, 1912. (Exchange.)

NEW SOUTH WALES. MINES DEPARTMENT. Mineral Resources. No. 15. Sydney, 1912. (Exchange.)

NOUVEAUX MÉCANISMES ET NOUVELLES MÉTHODES POUR L'ESSAI DES MÉTAUX. By Pierre Breuil. Paris, 1910. (Purchase.)

PERMANENTE GASE UND INDUSTRIE GASE, SAUERSTOFF-OEFEN, SAUERSTOFF-BRENNER, GAS-ELEKTRISCHE OEFEN. By G. Mettler. Leipzig, n. d. (Purchase.)

LES POMPES CENTRIFUGES A HAUTE PRESSION. By Ch. Hanocq. Liége, 1909. (Purchase.)

PRÉCIS DE MÉTALLURGIE. By H. Pécheux. Paris, 1910. (Purchase.)

PRINCIPLES OF IRONFOUNDING. By A. H. Sexton and J. S. G. Primrose. London, 1911. (Purchase.)

QUEBEC. DEPARTMEMT OF COLONIZATION, MINES AND FISHERIES. Report on Mining Operations in the Province of Quebec. Quebec, 1912. (Exchange.)

RECORD OF THE FIRST SERIES OF THE BRITISH COAL DUST EXPERIMENTS CONDUCTED BY THE COMMITTEE APPOINTED BY THE MINING ASSOCIATION OF GREAT BRITAIN. A Record of the Experiments carried out during 1908 and 1909, at the Altofts Experiments Station. London, 1910. (Purchase.)

DAS RETTUNGSWESEN IM BERGBAU. Seine Technik und gesetzliche Regelung im In-und Auslande. By O. Pütz. Freiberg, 1912. (Purchase.)

SOUTH MIDLAND COAL OWNERS. Report of the Mine Rescue Experimental Committee. An Investigation of Self-contained Breathing Apparatus for use in Mines, 1910–11. Birmingham, 1911. (Purchase.)

SPARK SPECTRA OF THE METALS. By C. E. Gissing. London, 1910. (Purchase.)

STEEL SHEET PILING. Tables and Data on the Properties and Uses of Sections. Pittsburgh, 1912. (Gift of Carnegie Steel Co.)

SUBWAYS AND TUNNELS OF NEW YORK: METHODS AND COSTS. By G. H. Gilbert, L. I. Wightman, and W. L. Saunders. New York, 1912. (Gift of W. L. Saunders.)

[NOTE.—Mining engineers are aware that for many years the city of New York has been one of the chief mining-districts of the world, by reason of the vast system of subways and tunnels, involving almost all the operations of mining—sinking, driving, hoisting, pumping, ventilation, timbering, and masonry supports, etc., together with many operations and problems not presented in ordinary mines, yet likely to be encountered hereafter in similar undertakings.

It is an interesting reflection, though it comes too late to be practically useful, that a large part of the $1,000,000,000 which the rapid-transit system of New York will have cost when the plans already authorized shall have been executed, has been devoted to unwise and futile schemes, which aggravated the evils they were intended to cure. Man's triumphs over natural obstacles have been many and creditable ; but it is not always necessary or advisable to treat nature as an enemy ; and sometimes man's interference simply defeats nature's remedies. The narrow island of Manhattan, with its magnificent water-front on the North and East rivers, was naturally adapted to the shipping business, and there was no good reason for putting into the limited space at its southern end the Stock Exchange, the Sub-Treasury, banks, newspapers, mercantile stores, and offices, etc., which could have been better accommodated "up town." But nobody realized adequately the great future of the metropolis, and the "opportunism"—*i. e.*, the seeking of immediate relief without forecast of ultimate results—co-operated with the force of habit to bring about an almost hopeless conflict. These elements of the problem which were really temporary were assumed to be permanent—and worse than that, unchangeable. The ferries from New Jersey and Long Island delivered their hundreds of thousands daily in the down-town district, which became so crowded as to over-tax the scanty means of street-transportation. The natural cure was indicated by a tendency in many lines of business to move uptown; but this relief was greatly delayed, and practically nullified for the time being, by expensive means, in the form of the Brooklyn bridge and the elevated railroads first, and of subways afterwards, intended to enable the vast multitude residing in Jersey, Long Island, and the Bronx to travel swiftly to and from the congested region in which they were, by tradition and habit, doomed to do their daily work. But the facilities of the elevated railroads invited the immediate introduction of vertical railroads, known as elevators, and the construction of lofty office-buildings. The combined result of bridges, railroads, lifts, and sky-scrapers has been to multiply many times the population of lower Manhattan, and to make the crowds more dense than ever. The "rush-hour" is fast extending over nearly the whole day; and it is safe to say that, when everything has been completed which has been begun, in the way of relief, the evil will be greater than it was when the first mistaken remedy was proposed. Meanwhile, we have four or five times as many persons going, coming, and working without any additional street-room or light or air.

So much for our ingenious and costly devices to perpetuate an unnecessary concentration of population in lower Manhattan!

But these remarks do not apply to the work of the Pennsylvania Railroad and Long Island system—almost the only one of the New York rapid-transit enterprises which works with natural economic laws, instead of resisting them. For, in spite of the artificial encouragement given to the congestion of day-population down-town, the irresistible current of northward movement has been, all the time, making itself felt; and the Pennsylvania Terminal, reinforcing that movement, has produced already a mighty change, which is a real relief. Whoever will observe the banks, stores, and other buildings between 30th and 50th streets, will recognize this fact. The land upon which our Engineering building stands is already worth twice as much as was paid for it eight years ago. In a word, the center of population and business in Manhattan has been moved at last, as it should have been moved long ago; and the Pennsylvania R. R. Co. has put its immense investment where it will do the most good, and no harm.

But, whether wise or unwise, the subways and tunnels of New York, together with its elevated railroads, bridges, and high buildings, have furnished to engineers a school of practice, a field for inventive genius and a record of achievement scarcely paralleled elsewhere; and it was a happy thought on the part of Mr. Saunders and his associates (I name him first, although his name appears last upon the title-page, because I feel sure that he has inspired and directed the work, and that his associates will heartily indorse my tribute to him), to collect into accessible form the instructive material furnished by the underground work involved in these undertakings. This material is specially valuable because most of the work in question was performed with the aid of abundant capital and the highest engineering skill, and therefore its methods and costs offer standards of efficiency and economy by which similar work can be measured.

The scope of the enterprises described in this book is stated by the authors as follows:

"The tunnel and subway system serving the population centering in New York includes: one complete subway system connecting three boroughs; eight subaqueous tunnels under the East river; six sub-aqueous tunnels under the Hudson river to the mainland; two sub-aqueous tunnels under the Harlem river to the Bronx; the belt-line tunnels and the New York subway of the McAdoo system; and the Bergen Hill and crosstown tunnels of the Pennsylvania R. R.

"In the aggregate, these enterprises probably involve as much capital as the building of the Panama canal—and possibly even more. They have encountered at every stage obstacles stupendous in magnitude and difficulty, and calling for engineering methods beyond all precedent. They represent engineering- and contract-achievement of such vast importance that they mark a new era in construction work."

The book presents in successive chapters the topography, geological formation, and historical data; Haskin's original Hudson tunnel; the East river gas-tunnel; the Manhattan-Bronx and Brooklyn-Manhattan divisions of the New York subway; the use of compressed air in the subway construction; the costs of excavation in the New York subway; the Bergen Hill and North River tunnels of the Pennsylvania R. R.; the excavation for the terminal station; the crosstown and East River tunnels; the Belmont and Hudson-Manhattan tunnels; and the Hudson terminal station. Fourteen appendices give data concerning air-compressors of various types; compressed-air locomotives; rock-drills; explosives; pumps; caissons; and tables of constants, etc., useful to engineers. The volume constitutes a treasury of useful information, especially as to actual costs, for which engineers will be grateful.—R. W. R.]

SUMMARY OF ALLOYS. The Employment of Physical Chemistry in Metallographie. By Ernest Jänecke. N. p., n. d. (Purchase.)

SVERIGES GEOLOGISKA UNDERSÖKNING. Arsbok, 1910. (L'Institut Royal Géologique de Suède.) Stockholm, 1910–11.) (Exchange.)

TEXT-BOOK OF RAND METALLURGICAL PRACTICE. Vol. I. London, 1912. (Gift of W. R. Dowling and W. A. Caldecott.)

[NOTE.—This is the first of two volumes issued in the well-known series of "Scientific Text-Books" of Charles Griffin & Co., Ltd., of London. In saying that it "meets a long-felt want," I am not merely echoing a phrase familiar in the advertisements of publishers. It has been for years a source of regret to me, as secretary of a technical society and as a student of metallurgy, that the progressive and busy experts of the Rand were not able to furnish to their professional brethren current notes of their experience. The few papers which I have secured by personal appeal for the *Transactions* of the Institute have been so interesting and valuable as to increase my desire for more. This work, prepared by many experts, partly (though not wholly) consoles me for many previous disappointments. It would be almost worth while to wait for needed knowledge (whatever might be lost in the interim), if one were sure of getting at last such satisfaction as is here furnished.

This volume, prepared by a considerable number of leading experts, contains chapters on The Witwatersrand and its Mines (by Ralph Stokes); Sorting and Breaking (by James E. Thomas); Stamp-Milling (by G. O. Smart); Tube-Drilling (by W. R. Dowling); Treatment of Sand (by James E. Thomas); Slime (by H. A. White and W. R. Dowling); Precipitation, and Clean-up and Smelting (by E. H. Johnson); Assaying and Testing (by J. McArthur Johnston); Chemistry of Banket Ore-Treatment (by W. A. Caldecott); and Tables (compiled by Messrs. Caldecott and White).

This volume is complete in itself, and can be separately purchased. Volume II. is to contain, as thus far announced, chapters on the Design and Construction of Plants and the Transport of Material, by C. O. Schmitt, M. I. M. M., a Councilor of the South African Institute of Engineers.

The foregoing statement of the contents of the book before me leaves me no room for detailed criticism. It contains, perhaps, too much obsolete matter, and fails to describe some of the most recent improvements. But it is the best work on its subject; and its excellent typography, abundant, pertinent, and really helpful illustrations, useful bibliographies and foot-note references, and serviceable index combine to make it highly valuable—not to say necessary—to students and practitioners alike. Messrs. Dowling and Caldecott, two of the contributing authors, who have presented this volume to the Institute, deserve the thanks of their fellow-members. I am happy to recognize also among the authors Messrs. Thomas, Smart, and E. H. Johnson as members of the Institute.—R. W. R.]

THEORY AND PRACTICE OF ENAMELLING ON IRON AND STEEL, WITH HISTORIC NOTES ON THE USE OF ENAMEL. By Julius Grünwald, translated by H. H. Hodgson, London, 1909. (Purchase.)

TRAITÉ DE MÉTALLOGRAPHIE. By Felix Robin. Paris, 1912. (Purchase.)

TRAITÉ DE MÉTALLURGIE DU FER. Vols. I.–II. By Léon Gages. Paris, 1898. (Purchase.)

TUNGSTEN MINING INDUSTRY IN NEW SOUTH WALES. (Mineral Resources No. 15.) Sydney, 1912. (Exchange.)

U. S. BUREAU OF CORPORATIONS. Report of the Commissioner of Corporations on the Transportation of Petroleum, May 2, 1906. Washington, 1906. (Gift of Department of Commerce and Labor, Bureau of Corporations.)

U. S. GEOLOGICAL SURVEY. Bulletin No. 498. Washington, 1912. (Exchange.)

——— Professional Paper No. 77. Washington, 1912. (Exchange.)

——— Water Supply Paper No. 295. Washington, 1912. (Exchange.)

U. S. NATIONAL MUSEUM. Proceedings. Vol. 41. Washington, 1912. (Exchange)

UNIVERSITY OF LEEDS. Eighth Report, 1910–11. Leeds, 1911. (Exchange.)

UNIVERSITY OF WISCONSIN. Catalogue, 1911–12. Madison, 1912. (Exchange.)

THE WAIHI GOLD MINING CO., LTD. Report of the Directors and Statement of Accounts for year ended Dec. 31, 1911. London, 1912. (Gift of Waihi Gold Mining Co., Ltd.)

TRADE CATALOGUES.

BRISTOL CO., Waterbury, Conn.
Bristol recording-gauges. 3 pages.

S. C. DINSMORE, Detroit, Mich.
The Atwater bomb calorimeter. 12 pages.

INGERSOLL-RAND CO., New York, N. Y.
"Little David" pneumatic drills. 16 pages.

A. LESCHEN & SONS, St. Louis, Mo.
"Leschen's Hercules," July, 1912. A publication issued by the company, containing information of interest to the users of Leschen's products. 12 pages.

United Engineering Society Library.

EARTHWORK DIAGRAMS. By C. A. James. White Haven, Pa., 1912. (Gift of Author.)

JOSEPH DIXON, ONE OF THE WORLD MAKERS. By E. Hubbard. East Aurora, 1912. (Gift of Joseph Dixon Crucible Co.)

NEBRASKA STATE RAILWAY COMMISSION. Annual Report to the Governor, 4th, 1911. Omaha, 1911. (Gift of Nebraska State Railway Commission.)

PITTSBURGH (PA.) FLOOD COMMISSION. Report, 1911. Pittsburgh, 1911. (Gift of the Commission.)

RAILWAY ECONOMICS. A Collective Catalogue of Books in Fourteen American Libraries. Prepared by The Bureau of Railway Economics. Chicago, n. d. (Gift of W. P. Cutter.)

GIFT OF WILLIAM PAUL GERHARD.

LINDLEY, W. H. Anlagen zu dem generellen Projekt dor Kanalisation der Stadt Würzburg. N. p., n. d.

——— Barmen-Elberfeld Entwässerung. Denkschrift uber die Klärung der Abwässer, vom 14 Sept., 1901.

——— ——— Erläuterungs-Bericht zum Projekt einer Klärungs- anlage für die Abwässer von Barmen-Elberfeld auf dem Gute Buchenhofen. 1902.

——— ——— Reise-Bericht der zum studium der Klärung der Abwässer nach England entsandten Commission. Elberfeld, 1902.

——— Entwässerung der stadt Homburg v. d. Höhe. Frankfurt-am-Main, 1886.

——— Erläuterungs-Bericht zu dem generellen Projekt der Kanalisation der Stadt Würzburg, Apr. 25, 1901.

——— Lodz. Entwässerung. Generelles Projekt. Erläuterungs-Bericht nebst 5 and 8 anlagen. Oct., 1909.

——— Wasserversergung der Stadt Lodz. Erläuterungs-Bericht zum Projekt. Oct., 1909. Frankfurt a/M., 1911.

UN DEN VEREHRLICHEN STADTRATH DER GROSSHERZOGLICH BADISCHEN HAUPTSTADT. Frankfurt a/M., 1890.

MEMBERSHIP.

NEW MEMBERS.

The following list comprises the names of those persons elected as members who accepted election during the month of July, 1912:

Members.

Burcham, William D., Min. Engr.....................Shafter, Pricidio Co., Texas.
Carney, Frank D., Genl. Supt.......................Penna. Steel Co., Steelton, Pa.
Cooke, Frederick W., Jr., Secy. and Treas., Hardinge Conical Mill Co., 50 Church St., New York, N. Y.
Cunniff, Bernard, Min. Engr....208 Bank and Trust Bldg., Knoxville, Tenn.
Dale, William C., Min. Engr................Casilla 230, Copiapo, Chile, So. Am.
Doak, Samuel E., Min. Engr.............Princess Furnace Co., Glen Wilton, Va.
Earle, Theodore, Min. Engr.......................165 Broadway, New York, N. Y.
Farrant, James C., Engr.......... .Room 1174, 50 Church St., New York, N. Y.
Ferguson, Claude, Min. Engr..Caldwell, Idaho.
Friedlander, Otto A., Mine Sampler, 48 Mark Lane, London, E. C., England.
Gannon, Michael H., Genl. Supt., Idaho-Continental Mines, Port Hill, Idaho.
Holden, James O. E., Supt., Princeton Coal & Land Co., Ltd., Princeton, B. C., Canada.
Houser, John N., Mine Mgr., American Zinc, Lead & Smelting Co., Carterville, Mo.
Jordan, Harold S., Min. Engr...Campo Seco, Cal.
Leaning, Eugene H.................Genl. Mgr., Nav Aug Coal Co., Scranton, Pa.
McCreery, J. Harold, Engrg. Asst...................49 Wall St., New York, N. Y.
McIntire, Robert, Min. Engr...Keating Gold Mining Co., Radersburg, Mont.
Maben, J. C., Jr., Vice-Pres., Sloss-Sheffield Steel & Iron Co., Birmingham, Ala.
Marsh, Harry W., Chem. and Geol.....White Horse, Yukon Territory, Canada.
Munroe, Harold S., Min. Engr., Care Butte & Superior Copper Co., Butte, Mont.
Phalen, William C., Geol...........U. S. Geological Survey, Washington, D. C.
Portugal, Joseph H., Min. Engr.........................Keeler, Inyo County, Cal.
Pratt, Paul David, Min. Engr..Libby, Mont.
Randall, Charles A., Min. and Met. Engr., Care Nipissing Mining Co.,'Ltd., Cobalt, Ont., Canada.
Schofield, Walter D., Min. Engr......Mine Safety Car No. 2, Trinidad, Colo.
Smith, Reuben E., Min. Engr...........Care C. L. Smith, Vladivostok, Siberia.
Smith, Sydney, Min. Engr..Juneau, Alaska.
Van Siclen, Matthew, Min. Engr..................44 Pine St., New York, N. Y.
Wright, Clarence A., Min. and Met. Engr., U. S. Bureau of Mines, Washington, D. C.

CANDIDATES FOR MEMBERSHIP.

The following persons have been proposed during the month of July, 1912, for election as members of the Institute. Their names are published for the information of members and associates, from whom the Committee on Membership earnestly invites confidential communications, favorable or unfavorable, concerning these candidates. A sufficient period (varying in the discretion of the Committee, according to the residence of the candidate) will be allowed for the reception of such communications, before any action upon these names by the Committee. After the lapse of this period, the Committee will recommend action by the Council, which has the power of final election.

Members.

Charles John Adami, Rivermines, Mo.
Proposed by E. B. Kirby, Philip N. Moore, O. M. Bilharz.

George S. Bailey, Spokane, Wash.
Proposed by Glenville A. Collins, L. K. Armstrong, Henry Kehoe.

Walter Amberson Barnes, Tooele, Utah.
Proposed by J. F. Kemp, Henry Krumb, E. P. Jennings.

William Newton Best, New York, N. Y.
Proposed by Leonard Waldo, H. M. Howe, Bradley Stoughton, Richard Moldenke.

Morrow Chamberlain, Chattanooga, Tenn.
Proposed by Joseph W. Richards, Howard Eckfeldt, Henry S. Drinker.

Charles Augustus Heckscher de Saulles, New York, N. Y.
Proposed by George C. Stone, J. A. Van Mater, William A. Pomeroy.

William Carter Dickerman, New York, N. Y.
Proposed by Joseph W. Richards, Howard Eckfeldt, Henry S. Drinker.

Amos Ewing Gillespie, Josephine, Penna.
Proposed by H. M. Howe, Henry D. Hibbard, Bradley Stoughton.

Samuel Verner Hunnings, Schenectady, N. Y.
Proposed by H. A. Beale, Jr., George Thomas, 3d, Bradley Stoughton.

Harry Dill Kinney, Tooele, Utah.
Proposed by J. F. Kemp, Henry Krumb, E. P. Jennings.

Paul Klopstock, Kennedy, Nev.
Proposed by J. F. Orr, W. W. Mein, John H. Lidgerwood.

Karl Louis Landgrebe, Ensley, Ala.
Proposed by D. B. Meacham, D. T. Croxton, B. D. Quarrie.

Harvey William Lindhardt, Columbus, Ohio.
Proposed by R. H. Sweetser, J. E. Thropp, Jr., J. H. Frantz.

Robert Alexander Lipscomb, Covington, Va.
Proposed by E. C. Means, F. U. Humbert, Frank Lyman.

Howard D. McLeod, Seattle, Wash.
Proposed by L. K. Armstrong, E. P. Spalding, J. V. Richards.

Edward Washburn Maynard, Senter, Mich.
Proposed by J. A. Capp, James B. Cooper, George L. Heath.

William R. Meyers, Princeton, Mich.
Proposed by Joseph W. Richards, Howard Eckfeldt, Henry S. Drinker.

James D. Mooney, Akron, Ohio.
Proposed by Henry M. Lane, Richard H. Vail, Clarence Carleton Semple.

Anton Curtiss Oberg, Hibbing, Minn.
Proposed by William Wearne, A. P. Silliman, O. B. Warren.

H. P. Parrock, Buffalo, N. Y.
Proposed by William R. Webster, Henry D. Hibbard, Bradley Stoughton.

Veryl Preston, New York, N. Y.
Proposed by Charles F. Rand, Bradley Stoughton, Jennings S. Cox, Jr.

Benjamin Franklin Roden, Marvel, Ala.
Proposed by Milton H. Fies, James Bonnyman, Richard Peters, Jr.

Fitz-William Sargent, Mahwah, N. J.
Proposed by Joseph W. Richards, Henry S. Drinker, Richard Moldenke.

Francis Lester Toy, Pittsburg, Pa.
Proposed by Joseph W. Richards, Howard Eckfeldt, Henry S. Drinker.

Lloyd Burton Treat, Gautemala, C. A.
Proposed by Henry S. Drinker, Howard Eckfeldt, Joseph W. Richards.

Associates.

John Summerfield Crowther, Jr., Cockeysville, Md.
Proposed by Joseph W. Richards, Howard Eckfeldt, Henry S. Drinker.

S. Kendrick Eshleman, Jr., New York, N. Y.
Proposed by Joseph W. Richards, Leonard Waldo, Henry S. Drinker.

John Haldeman Graybill, Pittsburg, Pa.
Proposed by Joseph W. Richards, Howard Eckfeldt, Henry S. Drinker.

Roderick Lee Johnston, Chrome, N. J.
Proposed by J. F. Kemp, Charles P. Berkey, Robert M. Draper.

Ernest Bowling Roeser, Kanawha Falls, W. Va.
Proposed by L. C. Crewe, Polk Tarwater, F. H. Clymer.

Changes of Address of Members.

The following changes of address of members have been received at the Secretary's office during the month of July, 1912. This list, together with the lists published in *Bulletin* Nos. 63 to 67, March to July, 1912, and the foregoing list of new members, therefore, supplements the annual list of members corrected to Feb. 1, 1912, and brings it up to the date of Aug. 1, 1912.

Adams, George I., Prof. of Geol............Pei Yang University, Tientsin, China.
Aldrich, Truman H., Jr..............615 Brown-Marx Bldg., Birmingham, Ala.
Aubury, Lewis E...Easton, Cal.
Baker, Charles J. S..........15 Eldon Rd., Kensington, London, W., England.
Banks, H. P., Care Mrs. G. W. Chase, 515 Julien St., Highland Park Sta.,
Los Angeles, Cal.
Bateson, Charles E. W..........................43 East 58th St., New York, N. Y.
Bowron, William M.......................Care *The Tradesman*, Chattanooga, Tenn.
Brayton, Corey C............... Room 408, 311 California St., San Francisco, Cal.
Chamberlin, John R., Mgr......Cinco de Mayo No. 10-D, Mexico City, Mexico.
Chase, Charles A..........................733 First Nat'l Bk. Bldg., Denver, Colo.
Chase, Edwin E......................................1450 William St., Denver, Colo.
Civretto, H. C...Instructed to hold all mail.
Clark, Clarence M.......................... ...321 Chestnut St., Philadelphia, Pa.
Clarke, Roy H.............................632 Old Natl. Bank Bldg., Spokane, Wash.
Cole, David..................................35 East University Ave., Catasauqua, Pa.
Cornell, Russell T..45 Cedar St., New York, N. Y.
Cornish, Harry L....................................54 Natoma St., San Francisco, Cal.
Coxe, E. J. D..308 S. 24th St., Philadelphia, Pa.
Crawford, John J.................................Bohemian Club, San Francisco, Cal.
Damon, George G........ ....5748 Kimbark Ave., Chicago, Ill.
Dodge, Edgar V.................... ....Box 93, Tonopah, Nev.
Dunn, Robert I..3336 37th Ave., So., Seattle, Wash.
Eldred, Byron E..............................230 Riverside Drive, New York, N. Y.
Elgin, William J., Supt., Winding Gulf Colliery Co., Winding Gulf, W. Va.
Eye, Clyde M., Mgr., Cinchena Mines..............................Chihuahua, Mexico.
Fairbairn, George......18 Welbeck Court, Kensington, London, W., England.
Farnum, Herbert C.........................6929 Hawthorne Blvd., Los Angeles, Cal.
Fishburn, Randolph E..................................P. O. Box 417, Tucson, Ariz.
Foster, Donald F., Care Prestea Block A, Ltd., via Secondi,
Gold Coast Colony, W. Africa.
Geoghegan, Robert H..................Rockfield, Blackrock, Co. Dublin, Ireland.
Gibbons, Charles A., Jr., Central Chile Copper Co., Ltd.,
Panulcillo, Coiqumbo, Chile, So. Amer.
Gould, Hubert J., Castlereagh Chambers, 10 Castlereagh St.,
Sydney, N. S. W., Aust.
Gouyard, G. M.......................................260 Convent Ave., New York, N. Y.
Guernsey, Forbes W., Canadian Pacific Ry. Co., Dept. of Natural Resources,
Bankhead, Alta., Canada.
Haas, Herbert..1428 Lafayette St., Alameda, Cal.
Hallett, S. I..847 Pearl St., Denver, Colo.
Hanckel, Christian...............Silverton Mines, Ltd., Silverton, B. C., Canada.
Hernandez, Marcos F..................Matamoros 212, Monterey, N. L., Mexico.
Hiller, Edward.......................................P. O. Box 884, Birmingham, Ala.
Hoffmann, Rudolf...........................Bergstrasse 948 I, Clausthal, Germany.
Hogarty, Barry.........................Care Garfield Smelting Co., Garfield, Utah.
Horwood, C. Baring.............................Tunstall Rectory, Suffolk, England.

Hudson, Albert W..........11 Cowles Road, Mosman, Sydney, N. S. W., Aust.
Humes, James, Cowichan Station, E. & N. Ry., Vancouver Island, B.C., Canada.
Ives, Glen P.......................................Cachinal, Taltal, Chile, So. Am.
Jessup, Albert B.....................................Jeddo, Luzerne Co., Pa.
Krejci, Milo W..................Anaconda Copper Mining Co., Great Falls, Mont.
Laird, George A.................................811 Mills Bldg., El Paso, Texas.
Langley, Seth S..............Room 321, I. W. Hellman Bldg., Los Angeles, Cal.
Leisenring, Albert C...Lost Creek, W. Va.
London, Clarence J...............................5525 Walnut St., Philadelphia, Pa.
Lyman, R. H., Mgr., Seneca Superior Silver Mines, Ltd., Cobalt, Ont., Canada.
Lyon, Dorsey A.............................Box 83, Oakland Station, Pittsburg, Pa.
McCormack, G. B.....................................P. O. Box 42, Birmingham, Ala.
McDevitt, James E...........................6910 Susquehanna St., Pittsburg, Pa.
McMaster, A. T. C...Sudbury, Ont., Canada.
Marshall, Emory M...Ludwig, Nev.
Merry, Henry G.............................956 Rockefeller Bldg., Cleveland, Ohio.
Miles, John B.....................................1011 Chestnut St., Philadelphia, Pa.
Mills, Charles E., Supt, Detroit Copper Mining Co. of Ariz., Miami, Ariz.
Moss, C. O...936 Union Oil Bldg., Los Angeles, Cal.
Newman, Maurice A..............................839 Mills Bldg., San Francisco, Cal.
Poole, William..............................School of Mines, Ballarat, Vic., Australia.
Powell, Frederick...........Care Seattle Const. & Dry Dock Co., Seattle, Wash.
Rattle, W J..8628 Euclid Ave., Cleveland, Ohio.
Read, Thomas T., *Mining and Scientific Press*, 420 Market St., San Francisco, Cal.
Richards, John V.......................1028 Old Nat'l Bank Bldg., Spokane, Wash.
Robbins, Hallet R.....................................607 Pioneer Bldg., Seattle, Wash.
Russell, William, Woodstock, Grove Crescent Rd.,
Kingston-on-Thames, England.
Sanderson, Henry S.....................................235 Foster Bldg., Denver, Colo.
Sawyer, Arthur H........................Care Baltic Mining Co., Redridge, Mich.
Saxman, Charles W....................................Instructed to hold all mail.
Schuettenhelm, John B...Garfield, Utah.
Scott, O. N.....................................Dominion Bond Bldg., Toronto, Ont., Canada.
Selbie, Charles C., Care Chosen Mining Co., Ko Sung Mines, Yeng Byen, Korea.
Steel, Alvin A...Colon, Neb.
Stockett, Lewis E., Dept. Natural Resources, Canadian Pacific Ry.,
Calgary, Alta., Canada.
Swanquist, Gustavus A., Care Butters Divisadero Co., Divisadero,
Salvador, C. A.
Thomas, John W...Saxton, Pa.
Verschoyle, William D., Genl. Mgr., Port of Bella Coola, Ltd.,
617 Pacific Bldg., Vancouver, B. C., Can.
Vivian, George G., Met.,.............East Butte Copper Mining Co., Butte, Mont.
Warriner, S. D., Lehigh Coal & Navigation Co., 437 Chestnut St.,
Philadelphia, Pa.
Watson, Robert W...14 Wall St., New York, N. Y.
Williams, Ralph B., American Smelters Securities Co., Velardena, Dur., Mexico.

ADDRESSES OF MEMBERS AND ASSOCIATES WANTED.

Name.	Last Address of Record, from which Mail has been Returned.
Baxter, Francis K., Jr.,	423 Wells Fargo Bldg., San Francisco, Cal.
Cook, Edward H.,	Minas Birimoa, S. A, Birimoa, via Canelas, Dur., Mexico.
Danforth, A. H.,	Cotopaxi, Colo.
David, W. M.,	Care C. G. Hussey & Co., Second Ave., Pittsburg, Pa.
Fergusson, Hugh B.,	216 Loo Bldg., Vancouver, B. C., Canada.
Fitzgerald, Thomas F. M.,	211 Sharon Bldg., Salt Lake City, Utah.
Furness, James W.,	Coffee, Trinity Co., Cal.
Gardner, E. D.,	Forest Service, Missoula, Mont.
Gmelin, Ernest,	Apartado 57, Aguascalientes, Mexico.
Green, George G.,	133a Vermont Ave., Washington, D. C.

Haggen, Edward A., ... 220 Pemberton Blk., Victoria, B. C., Canada.
Hamfeldt, George K., ... Care de Wendel & Co., Hayingen, Lothringen, Germany.
Hicks, Charles M., ... Austinville, Va.
Higgins, Edwin, ... Randsburg, Cal.
Hilson, Cleaveland, ... Livingston, Mont.
Hollis, R. W., ... Silverton, Colo.
Johnson, Dion L., ... 325 Water St., Pittsburg, Pa.
Lampshire, John O., ... Vulture Mine, Wickenburg, Ariz.
Leavell, John H., ... Buffalo Mine, Cobalt, Ont., Canada.
Le Noir, Frank H., ... Box 16, Mt. Bullion, Cal.
McDougall, Wallace D., ... 20 Bedford Place, Russell Sq., London, Eng.
Mitinskij, Alexander N., ... V. O. 12, line 33, St. Petersburg. Russia.
Moore, Roy W., ... P. O. Box 48, Velasco, Tex.
Munroe, Martin, ... Bengal Coal Co., Murulidih, Mohada, B. N. Ry., Bengal, India.
Nelson, D. W. C., ... Baker City, Ore.
Nobs, Frederick W., ... Negociacion Minera Santa Maria de Guadalupe y Anexas, S. A., Minillas, Zac., Mexico.
Pearson, William R., ... 628 W. 114th St., New York, N. Y.
Perks, Harry B., ... 419 Board of Trade Bldg., Portland, Ore.
Peterson, Frank, ... H. W. Hellman Bldg., Los Angeles, Cal.
Rathborne, Merwyn R. W., . . . Amargosa, via Las Vegas, Nev.
Rhew, James W., Cia. Minera y Exploradora de Ventanas, S. A., Ventanas, Dur., Mex.
Sheldon, Waldo, ... Urique, Chih., Mexico.
Short, Frank R., ... Carson City, Nev.
Thornton, Edward T., ... Apartado 30, Matehuala, S. L. P., Mexico.
Twynam, Henry, . . . O. K. Copper Mine, Cairns, No. Queensland, Australia.
Watson, Ralph W., ... Calloo, Utah, Clifton Mail box.
Webster, Erastus H., ... Hotel Cosmopolita, Guadalajara, Jal., Mexico.
White, W. P. . . . . 10 Bernard St., Spokane, Wash.

NECROLOGY.

The deaths of the following members were reported to the Secretary's office during the month of July, 1912:

Date of Election.	Name.	Date of Decease.
1897.	†Church, Myron J.,	May 21, 1912.
1907.	*Eatherly, Adrian D.,	—— —, ——.
1905.	*Maffett, Robert R.,	April 9, 1912.

* Member. † Associate.

SUBJECT TO REVISION.

[TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS.]

The Treatment of Complex Silver-Ore at the Lucky-Tiger Mine, El Tigre, Sonora, Mexico.

BY D. L. H. FORBES, TORONTO, CANADA.

(New York Meeting, February, 1912.)

Introduction.

THE application of the cyanide process to complex sulphide ores for the extraction of their gold and silver is no longer a novelty, but, as an example showing modern tendencies in this direction, the following description of the new milling process at the Lucky-Tiger mine is offered:

The extraordinary richness of the ore in the upper workings of this mine made any thought of milling in its early days seem unnecessary. Its location in an almost inaccessible part of the rugged mountains just south of the Arizona boundary-line, and 60 miles from the town of Douglas, made the expense of bringing in machinery and supplies a very serious consideration. But, after the property had been shipping high-grade ore on mule-back for a few years, there remained in the mine a considerable tonnage of vein-matter that was too low in value to ship, but which it was thought could be milled near the mine at a profit. The first milling-experiment consisted in the installation of Nissen stamps in conjunction with concentrating-tables and Frue vanners, but, as the ore from the upper workings contained most of its silver in the form of cerargyrite, this type of mill was not successful.

The proximity of the mine to the great copper-concentrating mills of Cananea and Moctezuma naturally made the influence of copper-milling ideas felt in the company's next installation. At considerable expense a stage-crushing concentrating-mill, with a capacity of 100 tons per day, was built. This mill was modeled along the lines of the most approved practice in copper-concentrators at that time, the main idea being that crushing by stages in rolls was necessary in order to prevent losses from "sliming." Largely owing to the fact that, when the mill began operation, the mine produced ore with very little cerargyrite, and with the silver and gold intimately associated with such minerals as galena, zinc-blende, chalcopyrite, and iron pyrite, this method of concentration was profitable, in spite of the fact that the mill-tailings carried 0.1 oz. of gold and 15.0 oz. of silver per ton, and power for operating the machinery cost $275 per h-p-year.

In the early part of 1910 I installed a small experimental cyanide-plant for the purpose of testing the ore and mill-tailings. The results of experiments made in this plant were so satisfactory that it was decided to commence work at once on the construction of a 250-ton cyanide-plant, and to make such changes and additions to the existing works as would make possible the most economical milling of 175 tons of ore from the mine daily, in addition to 75 tons of old tailings from the dumps below the mill. This work was completed and the new process in operation by the end of June, 1911. A view of the mill and cyanide-plant is given in Fig. 1, and the general plan and sections in Figs. 2 and 3.

Character of the Ore.

The results of tests made in the experimental plant established the following facts in regard to the character of the ore:

(1) Very fine grinding is necessary in order to obtain a good extraction of the silver. Grinding samples of the tailings to pass a 150-mesh screen and other samples of the same material to pass a 200-mesh screen before cyanide treatment gave results in favor of the finer grinding, showing an increase in extraction of from 15 to 20 per cent.

(2) A high decomposition of cyanide was to be expected, but, by careful concentration previous to treatment with solution, the consumption could be kept down to from 4 to 5 lb. of sodium cyanide per ton of tailings.

(3) Ore from certain sections of the property was found to be more amenable to cyanide-treatment than that from other places. In general, it was found that the oxidized ore, which yielded only a small recovery by concentration alone, gave the best extraction by combined concentration and cyanide-treatment of any ore in the mine. The cyanide-consumption on the oxidized ore was less than that on the sulphide ore. The worst

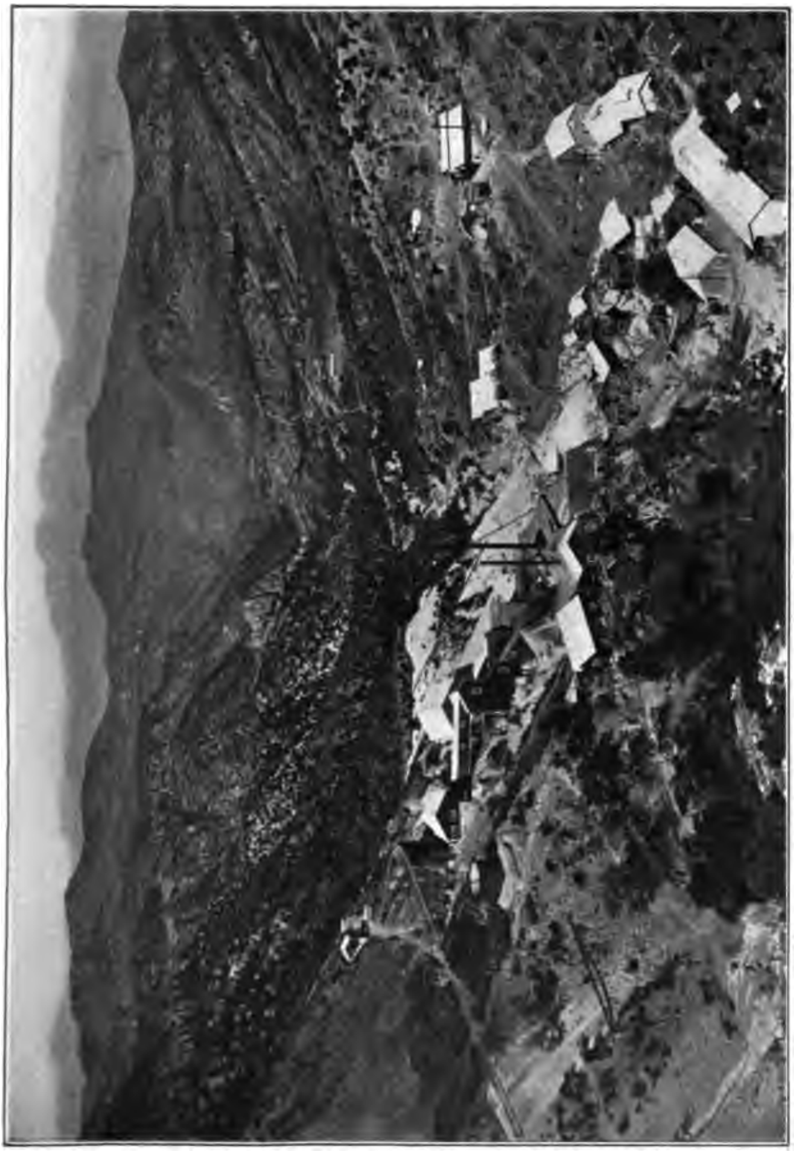


FIG. 1.—GENERAL VIEW OF MILL AND CYANIDE-PLANT OF THE TIGRE MINING CO., EL TIGRE, SONORA, MEXICO.

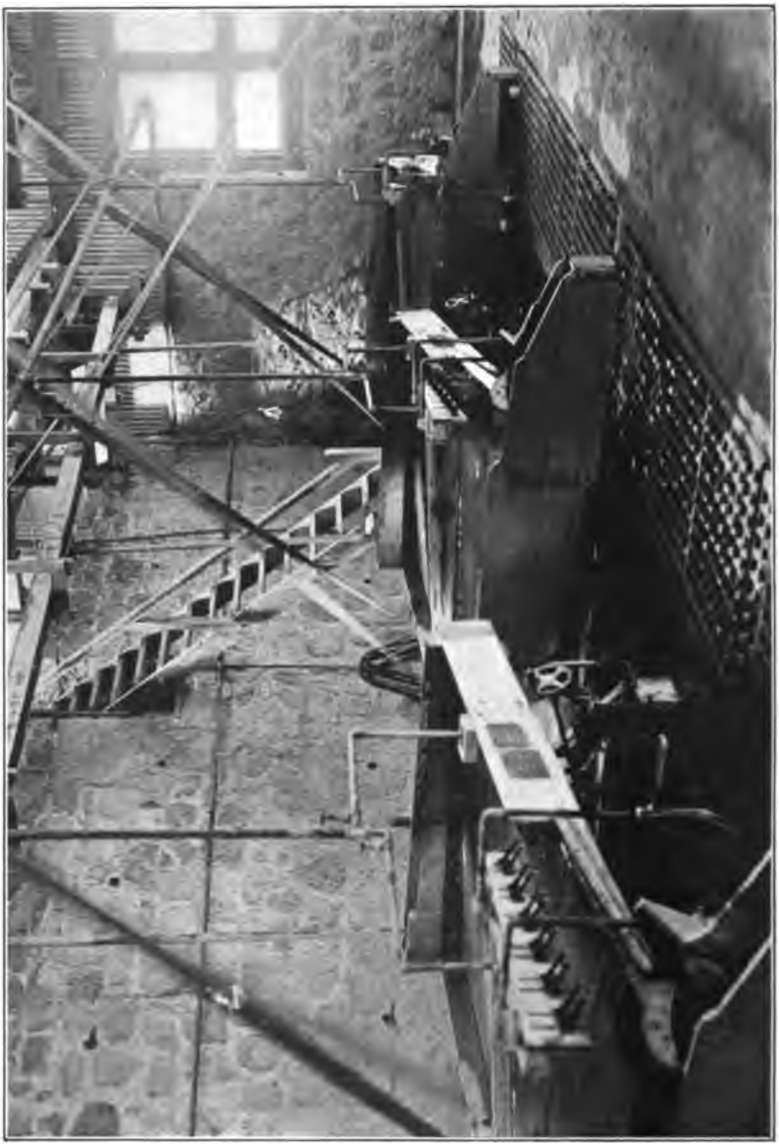


FIG. 6.—DEISTER SLIME-TABLES, NO. 2 MILL.

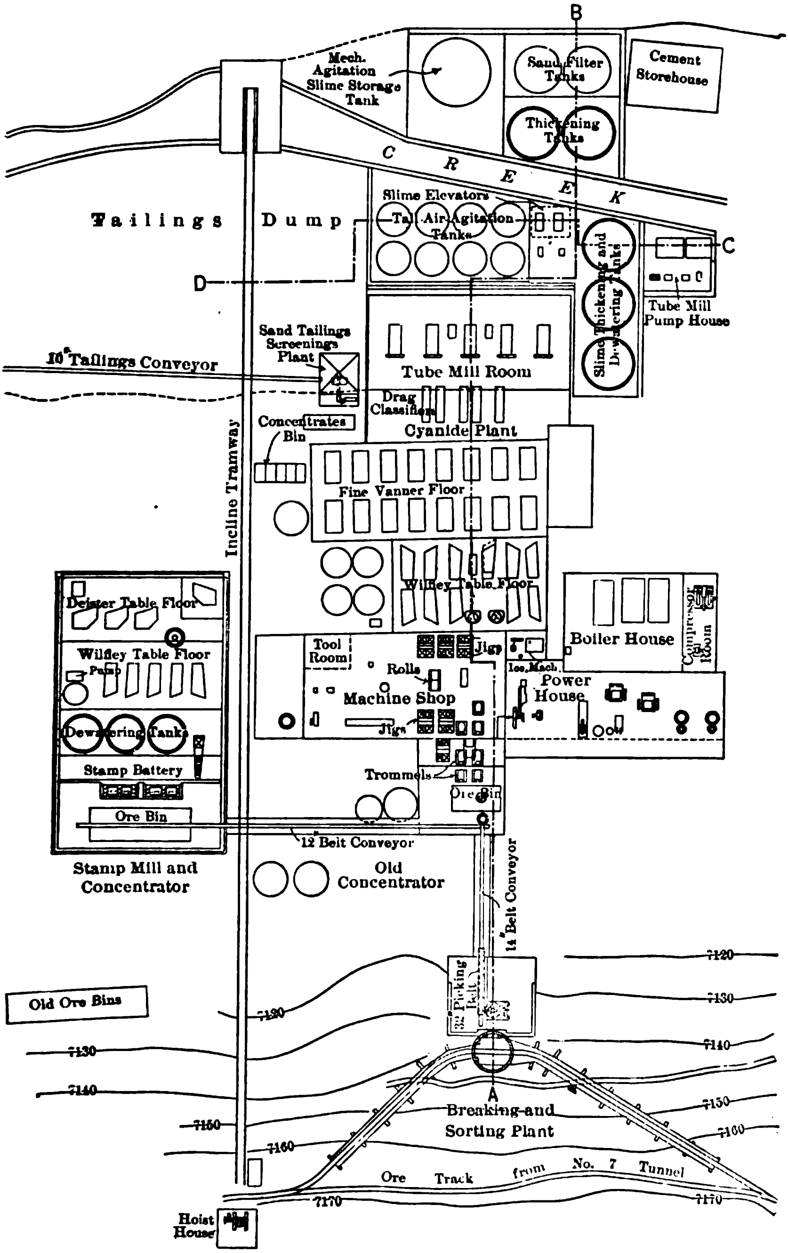


FIG. 2.—GENERAL PLAN OF CONCENTRATOR AND CYANIDE-PLANT.

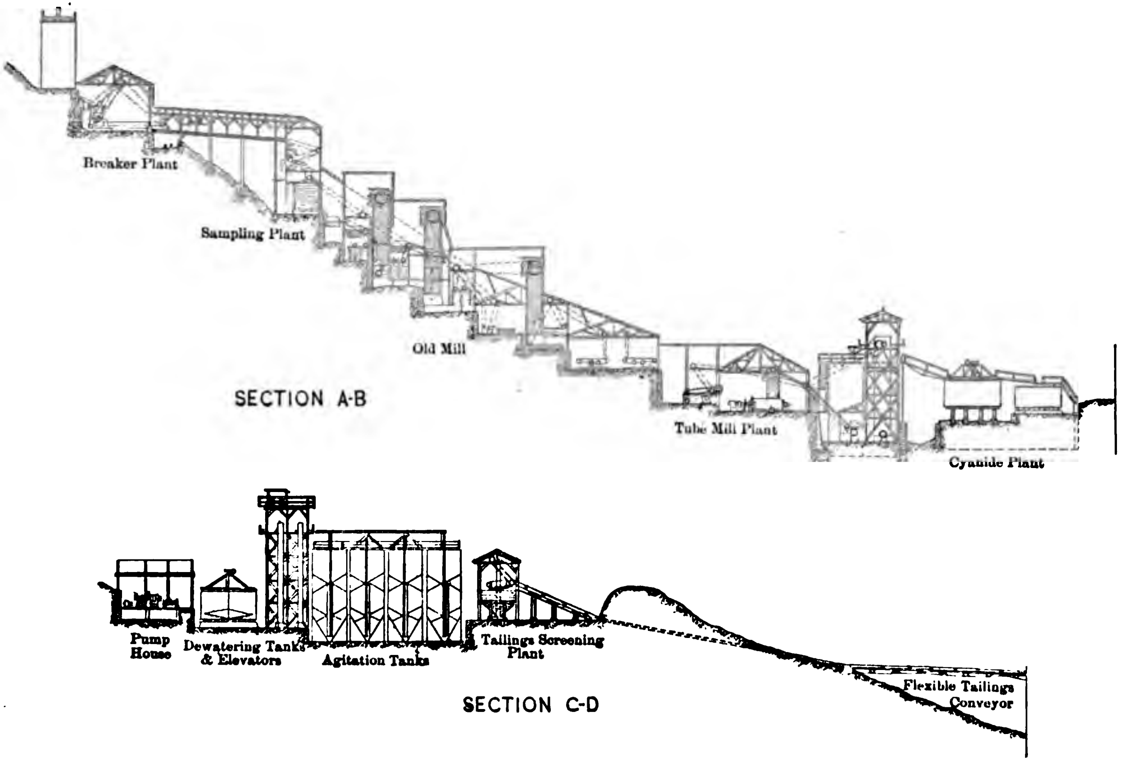


FIG. 3.—SECTIONAL VIEWS OF MILL AND CYANIDE-PLANT ON LINES A–B AND C–D OF FIG. 2

results, both in the matter of cyanide-consumption and in extraction, were obtained from the tailings from a certain class of sulphide ore that contained a high percentage of copper and was associated with massive zinc-blende and galena. This class of ore comes from a vein known as the "Sooy." The best results were obtained with the tailings of ore from the southern end of the property in the Gold Hill mine. The ore from the latter place is partly oxidized and is comparatively low in copper.

The average assay of 38,610 tons of ore milled in 1910 was: Au, 0.256 oz.; Ag, 33.568 oz.; CuO, 1.06, and PbO, 0.568 per cent.; Zn, present but not determined. A list of the minerals that have been recognized in the ore takes in the following: galena, zinc-blende, iron pyrite, chalcopyrite, bornite, argentite, stromeyerite, cerargyrite, hematite, pyrolusite, native gold, native silver, quartz, calcite, fluorspar, and alunite. Proustite is sometimes found, but its occurrence is rare. An aggregate which, to the naked eye, appears homogeneous, and has a silvery gray color, is frequently encountered in masses varying in size from several inches down to small grains, associated with pyrite, bornite, galena, and quartz. This was supposed to be tetrahedrite until a chemical analysis showed that there was no antimony present. The analysis gave:

	Per Cent.		Per Cent.
Ag,	18.5	As,	00.0
Au,	trace	Zn,	2.3
Cu,	35.7	Fe,	10.8
Pb,	7.3	S,	17.1
Bi,	00.0	Insoluble,	5.0
Sb,	00.0		96.7

Specific gravity, 5.65.

Specimens of this ore, submitted to Charles P. Berkey, of Columbia University, were pronounced by him, from microscopic evidence, to be aggregates. A polished surface of the ore under the microscope showed that it was an intergrowth of several simple metallic minerals which had developed a micrographic structure in solidification.

I found that in cyanide solution this particular ore acts very peculiarly. A sample ground so as to all pass through a 200-mesh screen was treated by successive air-agitations in fresh solutions of 0.04 per cent. of KCN, which carried also 0.5 per cent. of CaO and 0.05 per cent. of lead acetate. During the first 60 hr. of treatment, practically none of the silver went into solution, while the copper, on the other hand, was almost completely dissolved. At the end of 60 hr., solution of the silver commenced to take place, and was fairly rapid until the 100th hour. From 100 to 180 hr. the silver went into solution more slowly. At the end of 180 hr. the extraction of the silver was still only 87.86 per cent., while that of the copper was 97.73 per cent. The weight of potassium cyanide consumed or decomposed was actually greater than the weight of the sample. Naturally, the presence of any considerable quantity of this rich silver-copper compound would be very undesirable in the tailings when treated with cyanide solution, and, fortunately, it can be almost completely removed by careful table-concentration.

General Outline of the Milling Process.

The ore from the mine is hauled out by way of the No. 7, or lowest, tunnel in trains of 1-ton cars, by mules, and dumped into a 250-ton steel ore-bin at the top of the mill. An automatic feeder draws ore from the bin and drops it on a shaking-grizzly. The oversize from the grizzly falls into a gyratory breaker, and then is sorted on a picking-belt, where waste as well as high-grade ore is removed. This section of the plant has a capacity of 20 tons per hour, so that the required mill-tonnage may be crushed, sorted, weighed, sampled, and distributed to the two mills during the day shift. A complete flow-sheet of the milling and cyanide operation is given in Fig. 4.

Depending upon the distribution desired, from 50 to 56 per cent. of the crushed ore is sent to No. 1 mill, where it is crushed further by rolls, the sizing being done by trommels and hydraulic classifiers, and the concentrating by Harz jigs, Wilfley tables, and Frue vanners. The balance of the crushed ore from the breaker-plant is sent over a belt-conveyor to No. 2 mill, where the final crushing is done by twenty 1,200-lb. gravity stamps, the sizing by a Richards vortex type of hydraulic classifier, and the concentration by Wilfley sand-tables and Deister slimers. In the No. 2 mill the middlings from the Wilfley tables are reground in a 5-ft. grinding-pan, and join the slime middlings from the Deisters to be re-treated on a Deister slime-table. The results obtained from the No. 2 mill, with its simple but thorough scheme of concentration, have been considerably better than those obtained in the stage-crushing mill during the same periods, and will be referred to in detail further on in this paper.

On account of the rapid decomposition of cyanide if solution is allowed to be in contact with the sulphides of the Tigre ore, the concentration in both mills is done in water, sufficient lime being added to the crushed ore at the breaker-plant to aid the settling of slime, and to make the tailings alkaline before coming in contact with solution.

The tailings from both mills flow by gravity to the tube-mill section of the plant, where they are classified in two drag-classifiers and the material coarser than 200-mesh dewatered in two 5- by 14-ft. tube-mills. The product of these tube-mills is elevated, and joins the product of three other tube-mills for classification in solution by three drag-classifiers similar to the first two. The sand from the last three classifiers goes to the corresponding tube-mills, and in this manner a closed grinding-circuit is made from which only material of the required degree of fineness may escape. Old tailings from the dumps at the rate of 75 tons per day are delivered to drag-classifiers Nos. 3 and 4 after being thoroughly screened, sampled, weighed, and mixed with lime and solution.

The slime from No. 1 and No. 2 classifiers is dewatered by means of two 24- by 12-ft. Dorr slime-thickening tanks followed by vacuum filtration. The slime from the other classifiers, which is in solution, is thickened in a 24- by 12-ft. Dorr thickening-tank, and joins the dewatered slime for continuous treatment in a series of eight 15- by 40-ft. Pachuca tanks. At this stage the pulp has a consistency of 2 of liquid to 1 of solid by weight, and sodium cyanide is added at the first treatment-tank to bring the strength of solution up to 0.35 per cent. in terms of KCN. Protective alkalinity is maintained at from 1.5 to 2.0 lb. of CaO per ton of solution. The use of lead acetate was discontinued after the first six weeks of operation. In flowing through the series of treatment-tanks the slime receives the equivalent of 48 hr. treatment. From the last tank it is allowed to discharge into two 24- by 12-ft.

BREAKER AND SAMPLING-PLANT.

1 250-ton steel Ore-Bin, 15 ft. by 30 ft.

1 Revolving-drum Ore-Feeder, 24 in. by 27 in.

1 Shaking-Grizzly, 30 in. by 78 in., 1.5 in. spacing.

1 No. 4 Style "K" Allis-Chal. Breaker set to 1.5 in.

1 Sorting-Belt, 2.8 ft. long, 32 in. wide, 6-ply, ⅛ in. rubber cover, speed 50 ft. per min.

1 Blake-D. automatic Weighing-Machine.

1 Belt-Conveyor, width 14 in., length 87 in., speed 160 ft. per min., 5-ply, ⅛ in. cover.

1 Revolving-Cone Sampler and Distributor, 5 per cent. cut, sending 80 tons to

1 Belt-Conveyor, width 12 in., length 179 ft., 5-ply, ⅛ in. cover, speed 200 ft. per min.

1 Sample-Crusher, 7 in. by 10 in.

1 Snyder Sampler, 27 in. diam., 10 per cent. cut.

1 Sturtevant Roll-Jaw Sample-Crusher, 2 in. by 6 in.

1 Set Sample-Riffles cutting 10 per cent. sample.

1 Shaking-Grizzly, 30 in. by 78 in., ¾ in. spacing.

1 No. 2 Austin Gyratory Crusher, set to ¾ in.

1 Broken-Ore Bin, capacity 120 tons.

2 Eccentric and Arc Fine-Ore-Feeders.

2 Trommels, 4 ft. by 5 ft., ⅝ in. round holes.

OLD CONCENTRATING-PLANT.

2 Trommels, 4 ft. by 5 ft., ⅜ in. round holes.

1 Double-compartment Bull-Jig, 4-mesh wire screen, 107 2⅜-in. strokes per min.

1 D. C. Bull-Jig, 4-mesh, 107 1⅜-in. strokes per min.

2 Trommels, 4 ft. by 5 ft., $\frac{3}{16}$ in. round holes.

1 Cole Dewatering-Wheel.

1 Pair Roughing-Rolls, 16 in. by 36 in., Cole pattern.

2 D. C. Jigs, 23½ in. by 35¾ in., 6-mesh, 170 ¾-in. strokes per min.

1 Bucket-Elevator 39 ft. cc., 18 in. belt, 15 in. buckets.

2 D. C. Jigs, 23½ in. by 35¾ in., 8-mesh, 170 ⅝-in. strokes per min.

2 Trommels, 4 ft. by 5 ft., 1½-mm. round holes.

1 Cole Dewatering-Wheel.

1 Pair 16 in. by 36 in. Rolls.

1 Baffle-Screen removing chips from water.

2 Richards-Coggin Classifiers.

1 Bucket-Elevator, 42 ft. cc., 18 in. 10-ply belt, speed 360 ft. per min., buckets 15 in. by 7 in. by 6 in.

1 5-ft. Spitzkasten with baffle.

1 Richards-Coggin 3-spigot Classifier.

1 Three-compartment Jig, 10-mesh, 170 ⅝-in. strokes per min., middlings from hutch.

1 Dewatering Feed- and Storage-Box.

2 5-ft. Huntington Mills, ¾-mm. screens.

5 No. 5 Wilfley Tables, one for re-treating Jig-middlings.

4 16 ft. Redwood Settling-Tanks.

1 Bucket-Elevator, 35 ft. cc., 12 in. belt, 9 in. buckets.

6 No. 5 Wilfley Tables.

1 No. 5 Wilfley Table re-treating middlings.

1 9 in. by 10 in. Aldrich Triplex Pump, 47 R. P. M.

2 V-Bottom Slime-Thickeners.

1 Frenier Spiral Sand-Pump, 10 in. by 54 in.

2 Callow Pulp-Thickening Cones, 8 ft. diam.

16 6-ft. Frue Vanners.

Figures denote short tons per 24 hr. Ratios denote consistency of pulp by weight of solids and weight of liquids.

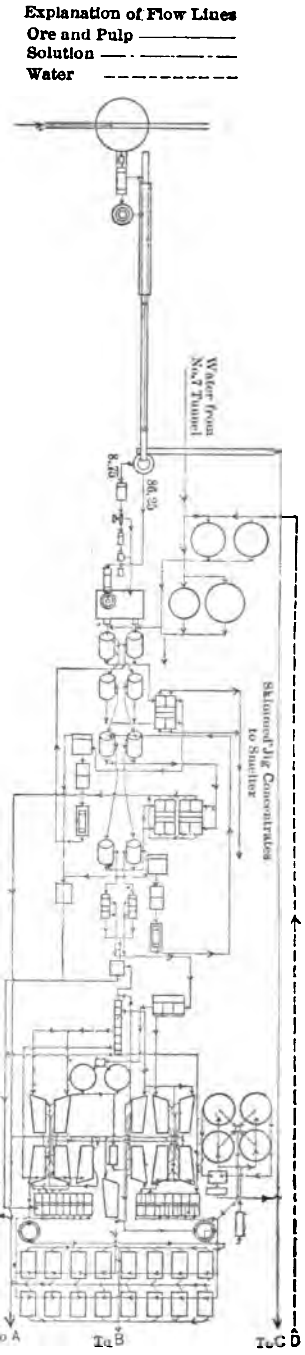
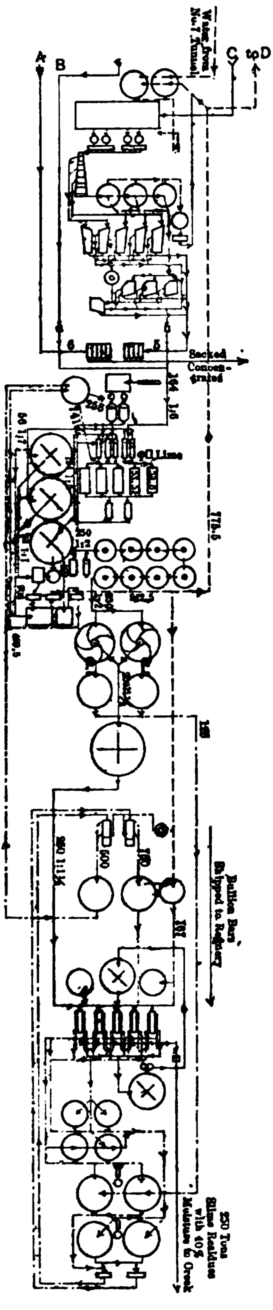


FIG. 4.—FLOW-SHEET FOR


A
B
C
to D
Water from No. 7 Tunnel
Sacked Concentrates
Lime
Bullion Bars Shipped to Refinery
250 Tons Slime Residue with 40% Moisture to Creek

MILLS AND CYANIDE-PLANT.

NEW STAMP-MILL AND CONCENTRATOR.

2 16 ft. Redwood Water-Tanks.
1 350-Ton Broken-Ore Bin fed by 12 in. Belt-Conveyor and Traveling-Tripper.
4 Suspended Challenge Ore-Feeders.
20 1,200-lb. Gravity Stamps, 20-mesh screens, 4 in. chuck-block. 102 7½-in. drops per min.
1 Hydraulic Vortex-Type Classifer.
3 16 ft. Conical-Bottom Dewatering-Tanks.
1 9 ft. by 8 ft. Conical-Bottom Return-Water Tank.
1 9 in. by 10 in. Triplex Return-Water Pump.
4 No. 6 Wilfley Tables, 240 R. P. M.
1 No. 5 Wilfley Table, 240 R. P. M.
1 5-ft. Grinding-Pan for middlings.
3 Deister Slime-Tables.
1 Deister Slime-Table re-treating middlings.
2 Scobey Tailings-Samplers.
2 Concentrates-Bins.
1 15 ft. by 10 ft. Steel Solution-Tank.

CYANIDE-PLANT.

1 System Flexible Tailings-Conveyor Belts for collecting dump-tailings.
2 Suspended Tailings-Feeders.
2 4 ft. by 5 ft. Trommels for screening and mixing tailings with solution.
5 42 in. by 16 ft. Drag-Classifiers.
1 Lime-Bin and feeder, 1½ tons p. d.
5 5 ft. by 14 ft. Wet-Grinding Tube-Mills.
2 24 ft. by 12 ft. Dorr Dewatering-Tanks, Steel.
2 Bucket-Elevators, 19 ft. cc., 18 in. belt, 15 in. buckets, speed 400 ft. per min.
1 24 ft. by 12 ft. Dorr Slime-Thickening Tank, steel.
1 Wilfley Dewatering Vac. Filter.
1 Mechanical Mixer for Slime-cakes
2 Bucket-Elevators, 39 ft. 9 in. cc., 18 in. belt.
8 15 ft. by 40 ft. Tall Steel Tanks for Treatment by submerged Air-Agitation.
3 Pumps—vacuum, water and solution.
2 Water proofed Concrete Sumps.
2 24 ft. by 14 ft. Tigre-type steel Pulp-Thickening Tanks.
2 20 ft. by 10 in. Sand-Filter Tanks, steel.
1 Tigre-type bottom-drive mechanical agitation Slime-Storage Tank, 30 ft. by 12 ft., steel.
2 Merrill Precipitation-Presses, one 20-frame for strong solution, other 30-frame for weak solution.
1 No. 125 Crucible "Steele-Harvey" Tilting Oil Furnace.
2 20 ft. by 10 ft. Steel Tanks for Barren Solution.
1 15 ft. by 10 ft. Steel Tank for Wash-Water.
1 20 ft. by 10 ft. Bottom-drive Mech. Agitating Tank for excess slime.
2 16 ft. by 10 ft. Redwood Conical-Bottom Water-Tanks—one for excess wash-water, other for cwt. mech'm.
5 5 ft. by 14 ft. "Parral"-Type Kelly Filter-Presses; each with 1,200 sq. ft. of Filter-area; spiral disintegrating-conveyor for breaking up and removing discharged cakes.
1 Scobey automatic Tailings-Sampler.
1 Air-Displacement Pump for returning excess slime.
1 20 ft. by 10 ft. Steel Agitating-Tank for excess slime.
4 15 ft. by 10 ft. steel tanks with sand-filter bottoms for clarifying solutions.
1 Zinc-Dust Feeder and Mixing-Cone.
2 20 ft. by 10 ft. Steel Tanks for strong pregnant solution.
1 Zinc-Dust Feeder and Mixing-Cone.
2 20 ft. by 10 ft. Steel Tanks for weak pregnant solution.
2 Aldrich 6 in. by 7 in. V. T. belt-driven solution-pumps.

CAPACITY.

Breaker and sampling-plant, 175 tons per 10 hr. Old concentrating-plant, 95 tons per 24 hr. New tamp-mill and concentrator, 80 tons per 24 hr. Cyanide-plant, 250 tons per 24 hr.

Tigre slime-thickening tanks. Here the pulp is thickened to a consistency of 1.5 to 1, and then stored in a 30- by 12-ft. mechanically-agitated slime-storage tank of the bottom-drive type. Five Parral-type Kelly filter-presses draw the thickened slime from this storage-tank at a pressure of 32 lb. per sq. inch.

Two kinds of solution, known as strong and weak, are drawn from the Kelly filters. These solutions are of nearly the same cyanide strength, but the strong solution is obtained during the formation of cakes on the filter-leaves, while the weak solution is obtained in the washing-period. The strong solution, to which is added that which is obtained at the two Tigre slime-thickeners, is partly precipitated with zinc-dust, and then returned to the plant for treating tailings as before. The weak solution forms a closed circuit, being precipitated completely with zinc-dust and then used again for washing the caked slime in the Kelly filters.

The precipitate, in spite of its high percentage in copper, is melted directly into bullion-bars in oil-fired tilting-furnaces. By a special method of fluxing and refining, which will be described later, the greater part of the copper and zinc in the precipitate is removed, and bars of from 800 to 850 fineness are obtained from precipitates of 60 per cent. average silver-content.

Bullion-bars and concentrates are sold, while the residues discharged from the filter-plant, which carry undissolved gold and silver to the value of from 80 cents to $1.10 per ton, are at present allowed to run to waste in the Tigre Cañon with from 42 to 45 per cent. of moisture. A dam is now being built below the plant to impound the residues.

Breaker-Plant and Sampling-Arrangements.

The run-of-mine ore is drawn from the 15- by 30-ft. steel ore-bin by a revolving-drum ore-feeder and falls on a 30- by 78-in. shaking-grizzly, the bars of which are spaced 1.5 in. apart and have a downward inclination of 14°. Owing to a device which gives a quick return-movement to the grizzly, the material on it is screened very thoroughly and made to run forward in a uniform stream to the gyratory breaker. Pieces of ore smaller than 1.5 in., which drop through the grizzly bars, are delivered from a spout on a 14-in. belt-conveyor that runs under and receives the discharge from the 32-in. sorting-belt. The sorting-belt takes the product of the No. 4 style K gyratory breaker. Three Chinese boys in 10 hr. are able to pick from the belt 1.5 tons of waste and 0.75 ton of high-grade ore. The average assays of waste and high-grade picked from the belt are as follows:

Waste—gold, trace; silver, 2.2 oz. per ton. High-grade—gold, 1.9; silver, 234.8 oz. per ton.

The percentage of recovery by sorting is as follows:

	Per Cent.
Of the gold in the run-of-mine ore, ...	3.7
Of the silver in the run-of-mine ore, ...	2.5
Of the value of gold and silver in ore, ...	2.7

The cost of sorting is about 4 cents per ton, of which labor is 3.2 cents, and power 0.7 cent.

All of the ore is weighed by means of a Blake-Dension machine as it is carried on the 14-in. conveying-belt to the concentrating-mills. This conveyor discharges the ore into a machine which distributes it in the desired proportion between the two mills, at the same time cutting a 5 per cent. sample. The distributing-and-sampling device, Fig. 5, consists of a cast-iron cone with a single discharge-spout that revolves about a vertical axis over a stationary cone with three compartments. Between the two large compartments is an adjustable partition which may be set to various positions corresponding to percentages of the total ore that it is desired to give to the No. 1 mill. From the compartment corresponding to No. 2 mill a discharge-spout delivers the material to a 12-in. conveying-belt, which carries it to the stamp-battery ore-bin. From the small sample-compartment the portion of ore cut for sampling is dropped into a 7- by 10-in. Blake crusher, which reduces the size of the largest pieces to 0.75-in. ring. A Snyder sampler placed below the crusher takes out 10 per cent. of the original sample, and the large pieces in this sample are crushed in a 2- by 6-in. Sturtevant roll-jaw crusher, after which the amount of the sample is reduced again by sets of riffle-boxes until about 40 lb. is obtained for sending to the assay-office. The rejected portions of ore at each step in the sampling drop into the broken-ore storage-bin of No. 1 concentrator.

Concentration in No. 1 *Mill.*

A shaking-grizzly with bars spaced 0.75 in. apart is placed below the distributing-machine and screens the ore sent to No. 1 mill. The oversize from this grizzly is reduced to pass a 1-in. ring by a No. 2 Austin gyratory crusher. The undersize from the grizzly and the product of the crusher are collected in a storage-bin of 120 tons capacity. Automatic feeders

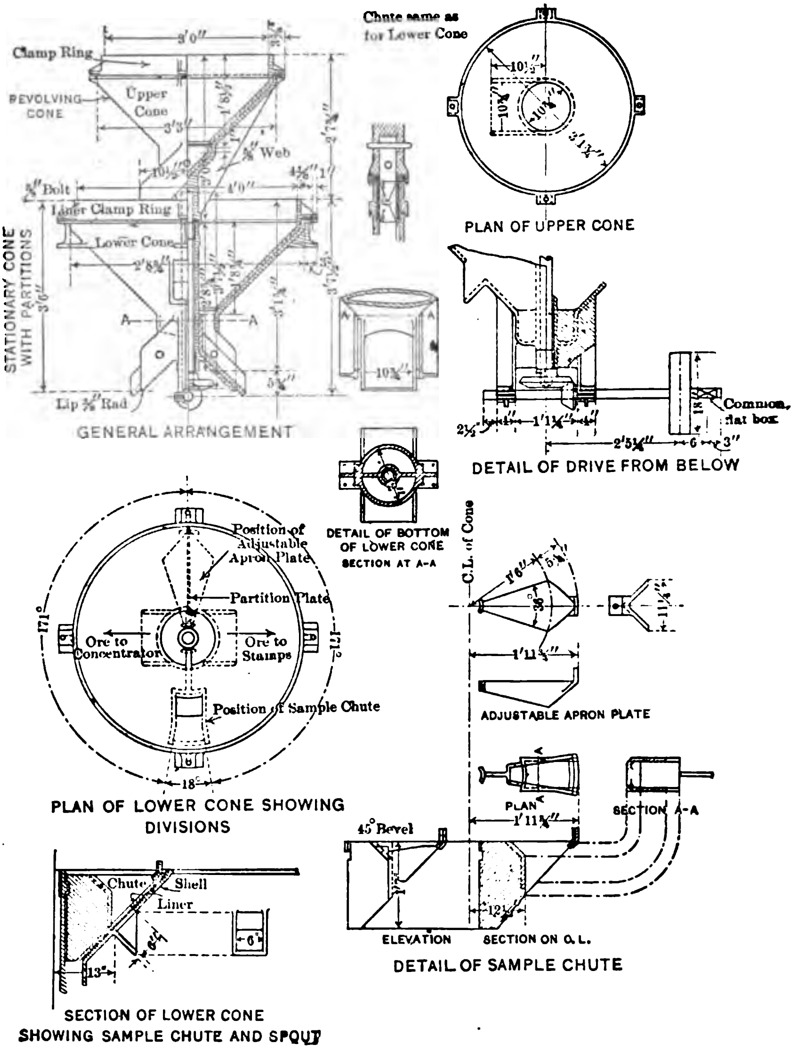


FIG. 5.—48-IN. DISTRIBUTING AND ORE-SAMPLING HOPPER.

draw the ore from this bin to the first trommels, where mill-water is added for the first time. The trommels are 4 ft. in diameter by 5 ft. in length, and have holes ⅝ in. in diameter punched in plates $\frac{5}{16}$ in. thick. The oversize from the first pair of trommels is carried with water in a launder to a double-compartment bull-jig of the Harz type, which makes 107 2⅜-in. strokes per minute, and uses a 4-mesh wire screen with a mineral bed in which concentrates are collected for periodical removal by an attendant with a shovel. The tailings from the jig are crushed in a pair of 16- by 36-in. rolls, and are then elevated and screened in a second pair of 4- by 5-ft. trommels.

The second trommels have ⅜-in. holes, and their product, consisting of pieces which range in size from ⅜ to ⅝ in., is delivered to a double-compartment bull-jig, similar to the first but having a 1⅜-in. stroke. The tailings from this jig join those from the first for crushing, and return to the second pair of trommels.

The concentrates from the bull-jigs amount to about 10 per cent. by weight of the total concentrates of the mill.

The third pair of trommels, which takes the undersize from the second pair, has $\frac{3}{16}$-in. holes, and its product is sent to four double-compartment Harz jigs, which make 170 strokes per minute. Two of these jigs work on a 0.75-in. stroke, while the others use a ⅝-in. stroke. The concentrates are drawn off continuously from iron hutch-doors, while the tailings are dewatered, crushed in a pair of 16- by 36-in. rolls, and then elevated to the third pair of trommels.

The fourth pair of trommels has 1.5-mm. holes and takes the undersize from the third pair. Its oversize is dropped to the 16- by 36-in. rolls, where it joins the tailings from the Harz jigs. Its undersize is sent to a pair of Richards-Coggin hydraulic classifiers, which sort out the coarser particles for concentration in a three-compartment Harz jig. This jig makes 170 ⅝-in. strokes per minute and produces middlings that are further concentrated on a Wilfley table, while its tailings are sent to one of two 5-ft. Huntington mills for regrinding through 0.75-mm. punched screens.

The overflow from the first classifiers is delivered to a 5-ft. *spitzkasten*. The overflow of the latter, consisting of slime fine enough for vanner-treatment, is sent to V-bottom slime-thickeners. The spigot-product of the *spitzkasten* is divided into four classes by means of a three-compartment Richards-Coggin classifier. Three of the classes consist of sand particles, which are treated on Wilfley tables. The fourth class is the slime overflow, which is sent to the V-bottom tanks for thickening previous to treatment on Frue vanners.

There are 12 No. 5 Wilfley tables, one of which is used for the re-treatment of the middlings from the others. Eleven of the tables make four products: slime, which is drawn off nearest the head of the table and sent to the Callow pulp-thickening cones previous to vanner-treatment; sand tailings, which are sent to the cyanide-plant; middlings, which are re-treated on the twelfth Wilfley table; and concentrates, which are carried away in shaking-launders and delivered to a dewatering storage-bin.

There are sixteen 6-ft. Frue vanners. The feed for them is thickened to about 1 to 4, solid to liquid ratio. The concentrates are partly removed from the belts by rollers on the under side, which are wound spirally with strips of old rubber belting. Tailings from the vanners join the sand tailings from the Wilfley tables and are sent to the cyanide-plant.

TABLE I.—*Wear of Roll-Shells at No.* 1 *Mill.*

	Coarse Rolls, 16 by 36 in.	Fine Rolls. 16 by 36 in.
Material	Midvale steel.	Midvale steel.
Service	Material $\frac{3}{16}$-to 1.5-in. size crushed to 0.25-in. ring.	Undersize from $\frac{3}{16}$-in. trommels crushed to 2 mm.
Original weight of each shell	1,417 lb.	1,417 lb.
Average weight when discarded	230 lb.	301 lb.
Per cent. efficiency wear of steel	84 per cent.	79 per cent.
Number of days in service	143	119
Total tons of ore treated	14,163.48	11,630.37
Cost of steel per ton of ore treated	$0.0324	$0.0376

NOTE.—The shells are rough machined with both inside and outside faces parallel, and are fastened to the roll-centers by hard-pine wedges $\frac{11}{16}$ in. thick. Alternating with the wooden wedges are iron wedges $\frac{3}{8}$ in. thick and 6 in. long. Shells fastened in this manner seldom slip, but usually break when the steel is about 0.5 in. thick.

TABLE II.—*Wear of Trommel-Screens at No. 1 Mill.*

	⅝-in. Trommels.	⅜-in. Trommels.	$\frac{3}{16}$-in. Trommels.	1.5-mm. Trommels.
Thickness of plates used	$\frac{5}{16}$ in.	0.25 in.	No. 10 gauge	No. 16 gauge
Number of units in use	4	8	8	12
Life of screens, months	7	7	6	2
Tons of ore screened	21,000	21,000	18,860	5,850

In the Huntington mills, both punched and woven-wire screens have been tried, with results in favor of the former: 1-mm. No. 26 gauge punched-plate screens, although their screening-capacity was somewhat less, outlasted the Tyler 0.75-mm. Ton-cap wire screens, and cost only $0.35 per unit, as against $0.96 for Ton-cap screens.

TABLE III.—*Wear of Harz Jig-Screens at No. 1 Mill.*

	Bull-Jigs.	Middle Jigs. No. 1.	Middle Jigs. No. 2.	Sand-Jigs.
Mesh of screen used	4	6	8	10
Size of wire in screen	No. 12	No. 14	No. 16	No. 20
Diameter of openings, inch	0.145	0.082	0.060	
Life with steel wire, days	39			
Life with brass wire, days	59	120	120	40

NOTE.—Steel wire has been found to be more economical than brass wire for the screens of the bull-jigs, on account of the cost of the steel wire being only one-third that of the brass.

Concentration in No. 2 Mill.

The broken ore for No. 2 mill is carried from the distributor to a 350-ton ore-bin by a 12-in. conveying-belt, and is evenly piled in the bin behind the stamps by an automatic traveling-tripper. From here the ore is drawn and fed to the stamps by four Challenge ore-feeders of the suspended type.

The stamps are each 1,200 lb. in weight when shoes are new. The dimensions of the principal parts are as follows:

Shoe,	9 in. by 9 in., made of chrome forged steel.
Die,	9.25 in. by 8 in., made of chrome forged steel.
Head,	9 in. by 20 in., made of chrome steel.
Stem,	3.75 in. by 14 ft., made of extra-refined steel.
Tappet,	9⅜ in. by 15.75 in., chrome steel.

When running at full capacity the stamps make 102 7.5-in. drops per minute, and use a 4-in. steel-faced chuck-block. The screens are 20-mesh Ludlow-Saylor rolled slot No. 25-wire screens. Under such conditions, and with 6 tons of water supplied to the mortars for each ton of ore, each stamp has a capacity of between 5 and 6 tons per day.

The mortars weigh 10,000 lb. and are of the narrow single-discharge type, El Oro pattern, and were specially manufactured by the Allis-Chalmers Co. in two sections with a vertical dove-tail longitudinal joint, in order to facilitate transportation in wagons from the railway to the mine. The battery-posts are set in cast-iron shoes which hold them securely to the concrete foundations. Rubber sheets 0.25 in. thick are placed under both the mortars and cast-iron post shoes. The 20 stamps and the 12-in. belt-conveyor are driven by a 75-h-p. slip-ring induction-motor.

A typical screen-test on the pulp discharged from the stamp. battery is as follows:

	Per Cent.
On 20-mesh screen,	0.5
On 40-mesh screen,	15.4
On 60-mesh screen,	15.7
On 80-mesh screen,	8.4
On 100-mesh screen,	3.0
On 200-mesh screen,	9.2
Through 200-mesh screen,	47.8
	100.0

The pulp from the stamps is classified into two grades of sand and one grade of slime by a Richards vortex-type hydraulic classifier which has four spigots. The first two spigots deliver coarse sand for treatment on two No. 6 Wilfley tables. The last two spigots separate the greater part of the fine sand particles from the slime for treatment on three No. 6 Wilfley tables. The slime overflow from the classifier is thickened in three cone-bottomed tanks, each of which is 16 ft. in diameter. The slime, at a consistency of 1 part of solid to 4 parts of water, is drawn by siphon-discharge from the settling-tanks and sent to three Deister slime-tables, Fig. 6. (It should be mentioned here, to avoid confusion, that in this paper the term "slime" is used to designate a suspension of ore particles in a liquid, where the size of the particles is such that they will pass through a standard 200-mesh woven-wire screen, irrespective of the physical character of the particles, which may be either crystalline or amorphous.)

The Wilfley tables, treating the coarse and fine sands from the hydraulic classifier, make three products—concentrates, middlings, and tailings. The middlings are reground in a Colorado Iron Works 5-ft. grinding-pan and join the middlings from the three Deister slime-tables for re-treatment on a fourth Deister table. The work done by the grinding-pan when working at a capacity of about 8 tons per day is shown by the following average screen-test:

Screen Size.	Feed. Per Cent.	Discharge. Per Cent.
On 40-mesh, ...	8.4	2.2
On 60-mesh, ...	13.4	10.0
On 80-mesh, ...	16.6	16.6
On 100-mesh, ...	12.0	15.0
On 200-mesh, ...	21.0	17.2
Through 200-mesh, ...	28.6	39.0
	100.0	100.0

It will be noted that there are no elevators in No. 2 mill, and that the scheme of treatment is very direct and simple. As a result, less power, fewer repairs, and smaller attendance are required for the treatment of the same tonnage of ore that is treated in No. 1 mill. It was not expected, however, that the No. 2 mill would give better recovery than the old mill from the Tigre ore; but, not only are the concentrates made by it of better grade than those from No. 1, but a better extraction is obtained, thus proving, for the case of this ore at least, that simplicity of treatment may aid recovery as well as reduce operating-cost.

TABLE IV.—*Screen-Tests Comparing Tailings and Concentrates from No. 1 and No. 2 Mills.*

Screen Size.	Tailings.		Table-Concentrates	
	No. 1 Mill.	No. 2 Mill.	No. 1 Mill.	No. 2 Mill.
	Per Cent.	Per Cent.	Per Cent.	Per Cent.
On 20-mesh..................	0.5	1.2	5.9	0.3
On 40-mesh..................	16.2	11.7	10.5	6.2
On 60-mesh..................	20.0	17.2	15.2	12.6
On 80-mesh..................	6.8	10.2	15.7	12.3
On 100-mesh..................	4.4	4.9	13.7	13.3
On 200-mesh..................	7.0	7.3	15.0	22.3
Through 200-mesh..........	45.1	47.5	24.0	33.0

The total average power, including lights, required for No. 1 mill, is 167.1 h-p., as compared with 86.2 h-p. for No. 2.

The costs per ton of ore milled are: No. 1 mill, $1.895, and No. 2 mill, $0.673 per ton.

The recoveries in the two mills, based on total value of the gold and silver produced in concentrates, compared with the value in the sorted mine-ore, are: No. 1 mill, 59.6, and No. 2 mill, 62.6 per cent.

TABLE V.—*Launder-Falls Used in No. 2 Mill.*

Location.	Size.		Dilution of Pulp.	Fall,
	Width.	Height.	S : L	Per Foot.
	Inch.	Inch.		Inch.
Stamps to classifier	8	8	1 : 6	7/8
Classified coarse sand to Wilfleys	6	6	1 : 3	7/8
Classified fine sand to Wilfleys	6	6	1 : 3	3/4
Slime to dewatering-tanks	8	8	1 : 20	1/8
Thickened slime to Deister tables	6	6	1 : 4	5/8
Sand tailings from Wilfley tables	8	8	1 : 5	1
Middlings from Wilfleys to pan	4	4	1 : 2	1
Concentrates from Wilfleys to bin	4	6	....	1 3/4
Slime tailings from Deister tables	6	6	1 : 7	3/8
Middlings from Deisters to 4th table	4	4	1 : 4	5/8
Concentrates from Deisters	4	6	....	1 1/8

Tube-Milling and Classification.

The equipment for regrinding the sand particles in the tailings consists of five 5-in. by 14-ft. wet-grinding tube-mills, made by the Power & Mining Machinery Co. These mills are equipped with spiral scoop-feeders, and are driven at the feed end by two sets of gears direct connected to individual motors by specially designed flexible leather-link couplings. The discharge ends are fitted with worm pebble-feed castings. Both ribbed liners and grating liners set on steel liner-plates are in use, with results in favor of the latter. The mills operate to best advantage when filled with pebbles to about 6 in. above the center-line. In spite of high first-cost, it has been found more economical to use only the best grade of imported flint pebbles, ranging in size from 2 to 4 in. in diameter. Quartz rock, obtained locally, was unsatisfactory, not only in the amount needed to keep the mills loaded, but also in efficiency of grinding and in wear on the linings. The sand fed to the tube-mills

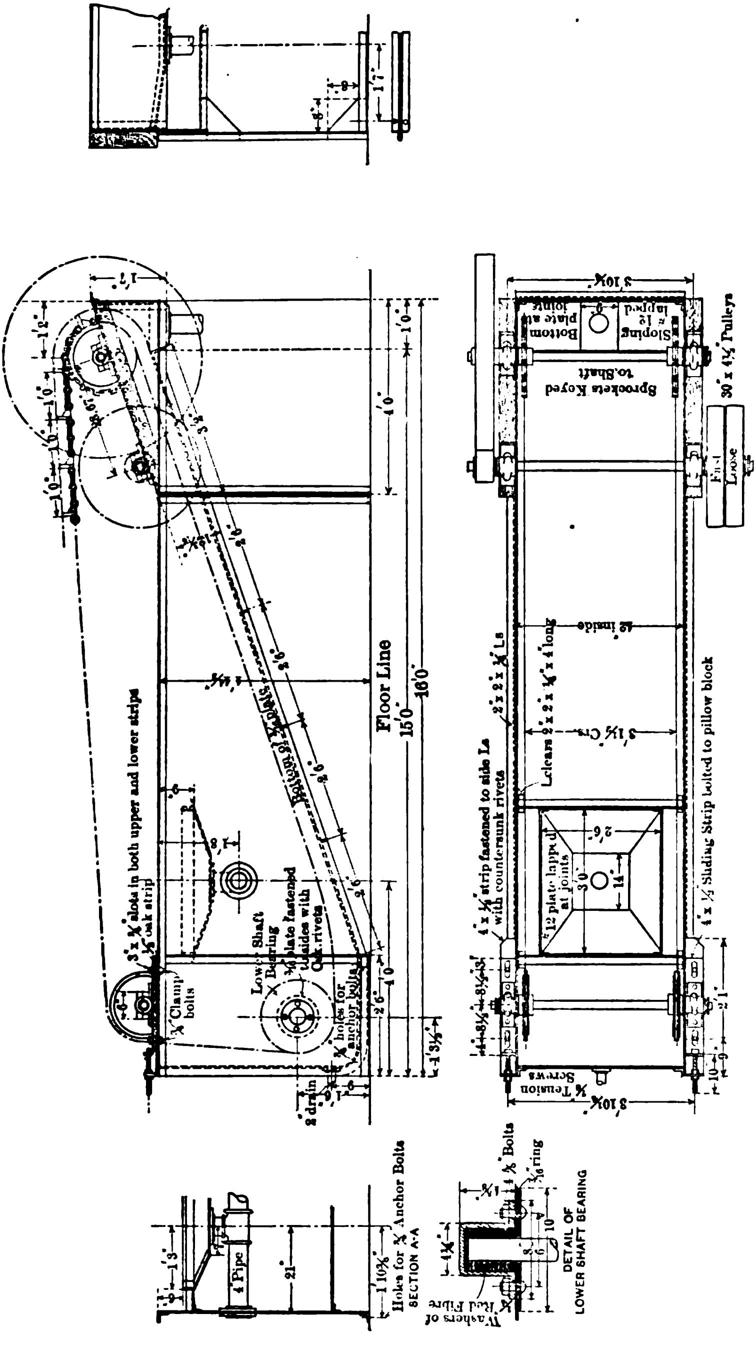


FIG. 7.—DETAIL OF 42-IN. DRAG CLASSIFIER.

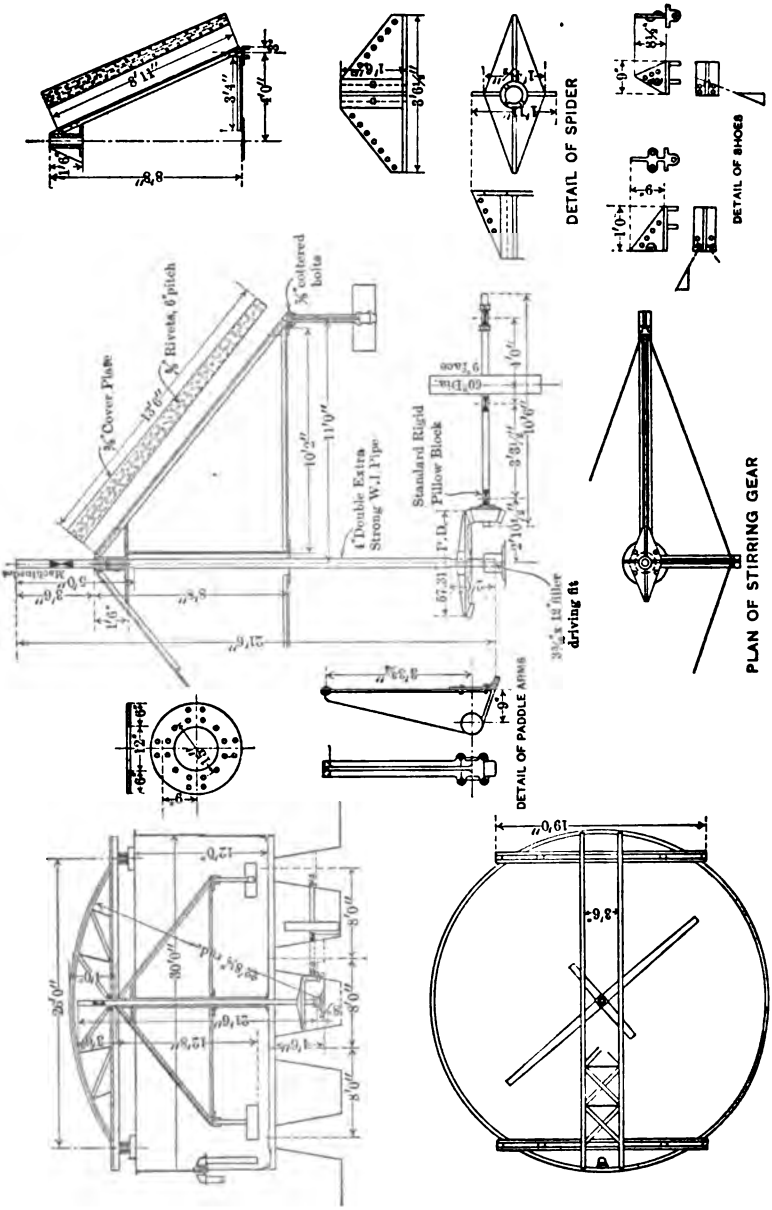


FIG. 8.—DETAIL OF MECHANICAL AGITATION SLIME-STORAGE TANK WITH BOTTOM DRIVE.

is maintained at a dilution of a little less than 1 part of solution to 1 part of sand by weight. Feed thicker than this deadens the blows of the pebbles in the mills and lowers the grinding-capacity, while a more dilute feed carries the sand through the mill before the particles have had sufficient grinding done upon them. The power consumed by each tube-mill ranges from 43 to 48 h-p., according to the height of the pebbles in the mill. While starting, each mill requires from 110 to 120 h-p. for a few seconds until the mill is brought up to its normal speed.

In order that the tube-mill plant may be capable of grinding the sand from 170 tons of concentrator-tailings, along with 75 tons of sand tailings from the dump, it is necessary to insure that the size of the sand particles from the concentrating-mills does not exceed 0.75 mm. When sand coarser than this limit is sent to the tube-mills, their capacity in product finer than 200-mesh diminishes very rapidly. Under normal conditions the slime delivered to the treatment-tanks has from 80 to 85 per cent. of material finer than 200-mesh and less than 5 per cent. coarser than 100-mesh.

The drag-classifiers (Fig. 7) which make the separation of sand from slime are in reality flight-conveyors, which elevate and dewater the particles of tailings that settle in steel boxes of special shape. It will be noticed from Fig. 8 that the overflow for the slime is such that the surface-velocity in the classifier is reduced to a minimum and so facilitates the settling of fine sand.

TABLE VI.—*Screen-Tests on Classifiers Nos.* 1 *and* 2, *Pulp-Dilution at* 1 : 6.

	On 100-mesh.	On 150-mesh.	On 200-mesh.	Through. 200-mesh.
	Per Cent.	Per Cent.	Per Cent.	Per Cent.
Feed	56.0	3.2	2.1	38.7
Slime product	....	....	9.5	90.5
Sand product	91.1	3.7	2.2	3.0

In connection with the tube-mill plant is a system of 10-in. flexible belt-conveyors for transporting tailings from the old dumps below the mill to a storage-bin, where they are thoroughly screened, mixed with lime, automatically sampled and weighed, and then made into a pulp with 0.15 per cent. of cyanide solution. The construction of these conveyors is such that the sand beneath them at any place may be excavated while the belt is running, and the framework sinks gradually into the new position. Loading-hoppers may be bolted to the framework at any point along the belt, and the tailings shoveled from under and from both sides of the conveyor. The operation of lowering usually proceeds from the tail end of the belt forward, and the trenches are made from 3 to 5 ft. deep by 12 ft. wide at each side. The head end of the first conveyor is driven by means of line-shafting and a universal coupling at the top of the storage-bin. When it is desired to change the alignment of this conveyor, its length is shortened to about 100 ft., and, as the work of excavation proceeds, new sections are added until the full length of 500 ft. is reached. Such a change is required only at intervals of six months or longer. At the present time only one other conveyor is used in connection with the first. This second conveyor, of a maximum length of 450 ft., is driven by a 5-h-p. motor, and discharges at any desired angle on the first. Eventually a third conveyor will be required to enable the complete removal of all the tailings in the dumps. Four Chinese shovelers load from 60 to 80 tons per 10 hr. on the belts, and they usually have two loading-hoppers in use at one time. A traveling-cableway with grab-buckets was planned at first for the excavation of the old dumps, but, after making a careful analysis of first-cost and operating-expense, it was concluded that a cableway installation would not be advantageous on the work of handling such a small tonnage of material per day. The cost of conveying and screening with the flexible conveyor arrangement is $0.11 per ton.

Dewatering.

The removal of mill-water from the tailings previous to treatment in solution is an obvious necessity in a cyanide-plant treating such rich silver-ore as that of El Tigre, for every ton of water introduced to the cyanide-plant has to displace a ton of weak solution somewhere, and even the weak solution here is stronger than the working-solution at most mills treating gold- or low-grade silver-ores. To avoid excessive waste of cyanide the dewatering of the tailings at El Tigre has to be very thorough.

No difficulty was experienced in the removal of water from the sand portion of the tailings, for, in the elevation of this material by the drag-classifiers, the sand is drained to from 20 to 25 per cent. of moisture by the time it reaches the discharge end of the machine, where solution sprays dilute it and wash it into the tube-mills.

In the case of the natural slime separated from the sand in the first two classifiers, a surprising amount of difficulty was encountered while trying to reduce the moisture-content to 40 per cent. It will be seen from the flow-sheet that two Dorr slime-thickening tanks, 24 ft. in diameter by 12 ft. in depth, were provided for the part dewatering of this slime. The rate of settlement was found to be very slow, however, owing to the amorphous character and extreme fineness of the material. The feed to the tanks consisted of a pulp which was normally at a dilution of 1 to 17, and, even when allowing the thickened slime to discharge at 1 to 4, clear water overflows were not obtained until accelerating-devices similar to those described by B. L. Gardner, which were used at the Sons of Gwalia mine, Australia, were placed in the tanks, and lime added to the ore at the breaker-plant in the proportion of 2 lb. of lime per ton of ore.

The final dewatering of the thickened slime is done in vacuum filters of local design, in which the cakes are thrown off and made into a pulp with solution of 0.15 per cent. of cyanide strength.

A curious characteristic of the natural slime in the ore at El Tigre is that, *per se*, it does not settle to a thicker pulp than 1 : 3 in 48 hr., even with the aid of lime or alum dissolved in the water. Lime, alum, cyanide, and similar salts have a marked effect in accelerating the settlement of the slime down to a point at about 1 : 6 dilution, after which further settlement is extremely slow, and finally reaches a point at between 1 : 3 and 1 : 4, where the slime seems to have reached what may be termed its critical condition, beyond which settlement is scarcely perceptible, even in periods of 24 hr. This critical condition of the slime is practically the same whether it has settled in clear water or in a solution of some salt.

Treatment.

The cyanide-treatment of the tailings really commences in the tube-mills, where the material is in contact with a solution of 0.15 per cent. of KCN. By the time that the slime has reached the first treatment-tank about 65 per cent. of the gold and about 50 per cent. of the silver in the tailings have been taken up by solution.

The treatment-tanks are of the Pachuca type,[1] 15 ft. in diameter by 40 ft. in height, and are arranged in a series of eight for continuous treatment. The total capacity of the eight tanks, with a hydraulic grade of 4 ft. between the first and last, is 560 tons of dry slime in a pulp, with dilution of 1 to 2; so that, when treating tailings at the rate of 250 tons per day, it is safe to say that all material in the tanks obtains at least 48 hr. of treatment. The chance of any material escaping from the series with less than two days' treatment is extremely small, owing to the number of tanks in the series; while the gain in time by not having to fill and discharge the tanks individually, coupled with the saving of corresponding expense in attendance, far more than offsets any real or supposed advantage that there might be in more positive control of the treatment when tanks are charged separately.

The slime in entering the first tank of the series falls on sodium cyanide cake in a perforated iron basket. In this way the solution is brought up to 0.25 per cent. in terms of KCN.

Air for agitation is compressed to 25 lb. per sq. in., and, to maintain a vigorous movement of slime in the tanks, from 80 to 100 cu. ft. of free air per minute per tank is required. The central air-lift pipe of each tank is 18 in. in diameter, and only extends to a point at two-thirds of the height of the tank, thus producing a submerged form of agitation, in which the air comes in contact with more particles of slime than in the usual form, in which the central pipe extends above the level of the slime.

For use in cases of emergency, high-pressure air-connections are supplied, by means of which air at 80 lb. pressure may be used inside the central tube, or on the outside by the aid of a temporary hose and air-pipe. As a further precaution, a centrifugal pump is placed under the tanks, and may draw off the contents of any tank in the series. For the purpose of dislodging any sand which might collect in the pipe-connections between the tanks, a high-pressure air-connection is furnished, which, in conjunction with the three-way iron cock at each of the connections, enables the operator to clean out the pipes whenever necessary.

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[1] *Bulletin* No. 62, February, 1912, p. 205.

The continuous-treatment system, as employed at El Tigre, has proved very satisfactory, and requires no attention except the occasional adjustment of valves by the shift-boss.

Slime-Thickening and Storage.

The normal dilution of the slime in the treatment-tanks is 1 to 2. For economical filtration, this pulp is too thin, and accordingly two 24- by 12-ft. slime-thickening tanks are used. These tanks give an almost perfectly clear solution overflow and discharge from the bottom a slime of about 1 to 1.5 dilution. Instead of using the Dorr type of continuous thickening mechanism for these tanks, spiral blades of sheet-steel supported from a central vertical shaft are used. This style of slime-thickening tank is illustrated in Fig. 8. The worm-gear wheel which drives the mechanism is keyed to the central shaft on a feather key, and permits the blades to be raised or lowered as desired. It will be noted from the illustration that the design of the blades is such that thickened slime anywhere on the bottom of the tank is continuously drawn in to the central discharge.

The thickened slime is stored in a 30- by 12-ft. tank, in which it is agitated by two pairs of steel arms with hinged iron paddles that are driven from beneath the tank, as shown in Fig. 9. The collar of this tank is 70 ft. above the center-line of the Kelly filters, and a mean gravity pressure for filtration of 32 lb. per sq. in. may be obtained at the filters while cakes are forming simultaneously in all five units.

Clarification of Solutions.

For economical and satisfactory precipitation of the gold and silver from cyanide solution it is an essential that the solution be perfectly clear, that is, that there be no solid matter in suspension previous to the addition of the zinc-dust. The overflowing solution from the thickening-tanks just described, while apparently clear to a casual observer, in reality still has a certain amount of very fine clay and silica particles in suspension. In order to clarify this and also the cloudy solutions

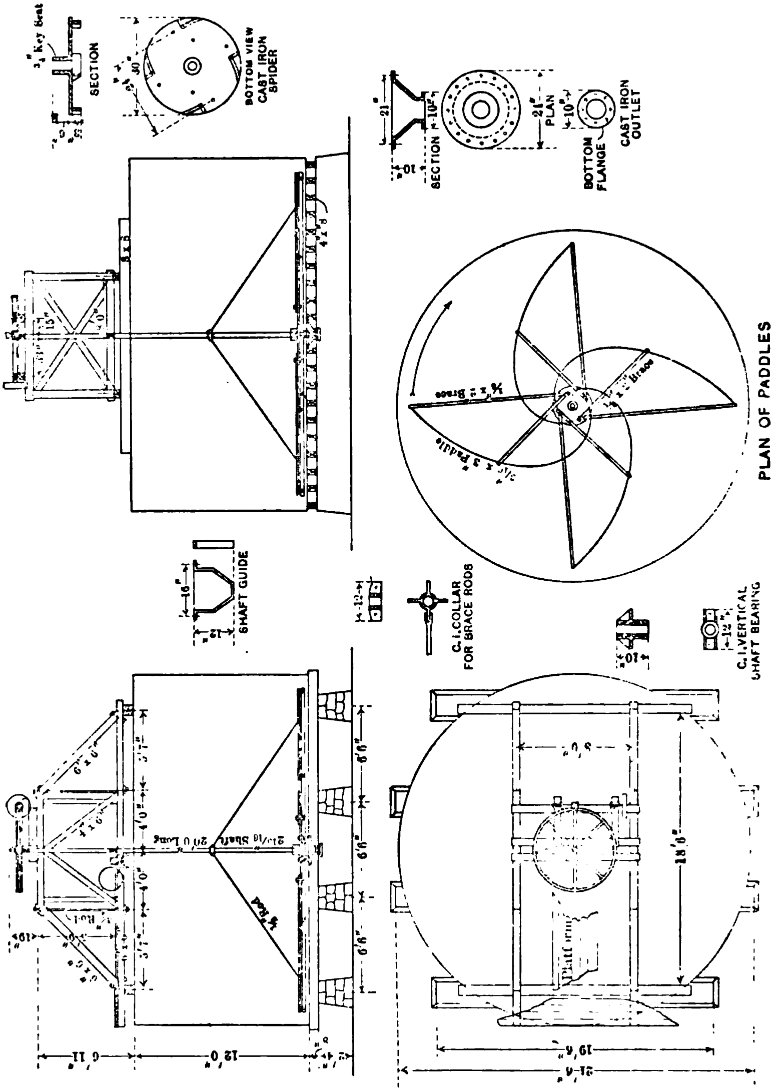

FIG. 9.—DETAILS OF TIGRE-TYPE THICKENING-TANK.

obtained at the filter-plant, sand-filter tanks of the usual type were tried at first, but were afterwards altered, and the sand-filter bottoms replaced by locally designed vertical leaves. Each leaf is 42 by 120 in. in size, and is made of grooved wooden slats placed above a perforated iron pipe and inclosed between two sheets of canvas that are sealed all around by P. & B. paint and a bolted iron frame. In this way a perfectly tight filter-leaf is obtained which has no stitching or nail-holes exposed at any part of the surface. The leaves are supported on channel-irons with their under sides 12 in. above the bottom of the tank, thus permitting the greater part of the sediment to accumulate on the tank floor. The leaves are spaced 18 in. apart so that, once in six weeks, the cake which has accumulated may be removed by hand with wooden scrapers, and the canvas scrubbed with a weak solution of hydrochloric acid. The two upper clarifying-tanks are 20 ft. in diameter by 10 ft. high, and the sediment on the bottoms may be washed out by a hose into the 30- by 12-ft. storage-tank. At the filter-plant there are four 15- by 10-ft. clarifying-tanks, which are operated in a similar manner except that, when it is desired to clean them, the solution is drained off thoroughly, a wash of water given, and the sediment then allowed to run to waste.

Filtration.

The choice of the Kelly type of slime-filters (a photographic view of a Kelly filter is given in the paper of W. P. Lass on the cyanide-plant at the Treadwell mine, Alaska)[2] for the Tigre cyanide-plant depended not so much upon maximum efficiency in filtration as upon conservation of water. Owing to the location of the plant in a section of the country which is periodically subjected to prolonged dry seasons, and to the fact that the supply of mill-water is derived solely from the mine and varies according to the rainfall at different seasons, it was imperative to insure economy of water in designing the plant. Only after a careful survey of and numerous tests with various types of filters which would operate successfully on the clay slimes of El Tigre, it was decided to install five units of the Parral type of Kelly presses. After solving certain initial mechanical difficulties inherent in filters of this sort, the Kelly plant has proved fairly satisfactory and gives excellent metallurgical results, though at high operating-cost.

The filter consists of a number of vertical leaves attached to a movable carriage that runs in and out of an inclined sheet-steel cylinder. The movable head of the cylinder is made to seal the lower end hermetically when the carriage is run in for the filtering-operation. When the carriage is run out with the cages on the leaves, clear water and compressed air are forced from the inside of the leaves outward, and throw off the cakes into a large steel-lined trough, at the bottom of which a spiral ribbon-conveyor breaks up the cakes of filtered residue and pushes it forward to the discharge-launder. The complete removal of all slime that may adhere to the leaves, after discharging with water and air from the inside, is effected by directing a small stream of water at high pressure through a hose between the leaves.

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[2] *Bulletin* No. 62, February, 1912, p. 201.

The slime-cake, after air-drying on the leaves, contains only from 24 to 28 per cent. of moisture, but, at the present time, water is added during the discharge until there is about 45 per cent., at which point the slime may be conveniently carried off in a launder and sent to waste. It is probable, however, that an attempt will soon be made to remove the slime residue from the plant at about 30 per cent. of moisture and stack it by means of a flight-conveyor.

Each filter has 13 leaves, composed of an iron-pipe frame, with sewed canvas, inclosing vertical grooved wooden slats. The total filter-area of each press is approximately 1,200 sq. ft., and the capacity 50 tons of dry slime per day. The normal cycle of filtering-operations is:

	Time. Minutes.	Pressure. Pounds.
Filling with slime,	4	
Forming cake on the leaves,	20	10–32
Discharging excess slime and filling with solution,	6	4
Washing the cakes with barren solution,	30	25–34
Discharging excess solution and filling with water,	6	4
Washing with water,	4	25
Blowing back excess water and drying cakes with air,	5	10–15
Discharging cakes and returning carriage,	40–60	

Total time of cycle, from 1 hr. 55 min. to 2 hr. 15 min.

In order to prevent a building-up of dissolved gold and silver in the excess solution used for washing, this solution is run into the filters instead of slime once every shift, and in this way the efficiency of the cyanide wash is made dependent only upon the skill of the operator and the completeness of the precipitation of the weak solution.

The excess slime is discharged by gravity from the presses to a tank, from which it is returned to the filling slime-storage tank by two Frenier sand-pumps in series. Excess wash-solution and water are returned to their respective filling-tanks by compressed air admitted to the filter-shells.

All five presses are operated as one unit from a single point of control, where the operator has the valve-levers conveniently arranged.

Once in each month a wash of dilute hydrochloric acid is given the leaves by cutting out one press from the rest and pumping acid through the leaves for about 30 min. The strength of the acid is determined by testing the filtrate. Commercial hydrochloric acid is added to the acid-storage tank until the filtrate gives a distinct red coloration to the litmus test-paper.

During the first 3 min. of filtering or washing, the effluent solutions are usually cloudy, and, at such times, are sent to clarifying-tanks before being precipitated with zinc-dust.

Precipitation.

The Merrill process of zinc-dust precipitation is used. The zinc-dust comes in zinc-lined cases, and great care is taken to prevent oxidation. The feeders are of the improved hopper-and-worm type, which delivers a measured quantity of dust continuously to a small multiplex tube-mill, in which the dust is thoroughly emulsified with a stream of pregnant solution which comes from the upper clarifying-tanks. The emulsion flows from the little tube-mill continuously through a rubber hose placed in a vertical pipe connected with the suction of a 6- by 7-in. vertical triplex belt-driven Aldrich pump, which elevates the solution to the Merrill precipitation-press placed in the melting-room. The use of the little tube-mill for emulsifying the zinc-dust, when precipitating silver solutions, insures a uniform feed of zinc-dust, brightens and subdivides the particles of zinc, and prevents trouble with the pumps from formation of zinc-cake on the valve-seats.

Strong solution is precipitated at the rate of 500 tons per day in a triangular 52-in. 20-frame press, the frames being 3 in. deep. Weak solution is precipitated at the rate of from 600 to 750 tons per day in a similar press that has 30 frames. Since the strong solution is returned for the treatment of fresh tailings, complete precipitation is not required, and so a considerable saving in zinc-dust may be obtained. The weak solution, however, which is used for washing in the filter-plant, is precipitated as completely as practicable. The strong-solution press is cleaned up every week, while the weak-solution press is usually run for two weeks. Both presses have the discharge-cocks at the top in order to maintain a constant level of solution. The concrete floor under the presses drains to a small sump, from which all floor-washings are picked up and returned to the presses by means of a small auxiliary pump and motor, which are in use only about 5 min. each day. For clean-up a large sheet-steel tray on wheels is run under the press, and the precipitate, after being partly dried with air in the press, is scraped out of the frames and allowed to fall into the tray.

The strong solution heads average 0.02 oz. of gold and 4.50 oz. of silver per ton. A typical analysis of the precipitate from the strong solution is: Au, 0.31; Ag, 61.96; Cu, 8.20; CuO, 3.01; Zn, 1.70; ZnO, 5.85; SiO_2, 4.80; Al_2O_3, 4.50; Fe_2O_3, 1.50; CaO, 2.12; undetermined, 6.05; total, 100.00 per cent.

The heads of the weak solution average 0.016 oz. of gold and 3.2 oz. of silver per ton. The barren weak solution averages 0.001 oz. of gold and 0.16 oz. of silver for the entire period between clean-ups, being higher at the beginning and lower at the end of the time. A typical analysis of the precipitate from the weak solution is: Au, 0.20; Ag, 41.82; Cu, 19.80; CuO, 4.01; Zn, 0.90; ZnO, 10.00; SiO_2, 4.30; Al_2O_3, 3.70; Fe_2O_3, 1.30; CaO, 2.16; undetermined, 11.81; total, 100.00 per cent.

The zinc-dust used for precipitation contains 94.7 per cent. total zinc, of which 92.7 per cent. is in the metallic state.

Melting and Refining.

The precipitate taken from the presses is dried in an electrically-heated car, which has a series of superimposed shallow pans, under which the resistance-coils are placed. On account of the large amount of moisture given off from the precipitate while drying, the resistance-wires have to be protected by an asbestos-paint covering in order to prevent corrosion. The car has a capacity for drying 1,000 lb. of precipitate in 10 hr. and takes 4.5 kw. of power.

Melting is done in two No. 125 crucible Steele-Harvey oil-burning furnaces of the tilting type. The object of the melting is to fuse the precious metals and to remove as much of the copper and zinc as possible in a slag which will also carry off the baser impurities, such as alumina, silica, lime, and iron. To effect this purpose, fluxes are added to the precipitate in such proportions that the oxides of copper and zinc find sufficient finely-powdered silica present to unite with it and form subsilicates; the oxides of iron, aluminum, and calcium find sufficient borax and silica to dissolve or unite with them; while a small amount of sodium bicarbonate and a little fluorspar are added to make the slag fluid.

Flux for Strong-Solution Precipitate.

Analysis of Precipitate.	Silica in Parts Per 100 to be Added.	Borax in Parts Per 100 to be Added.	Fluorspar.	Soda Bicarbonate.
Per Cent.				
Au......... 0.30				
Ag.........63.13				
Cu......... 1.5				
CuO....... 3.14	Plus, 2.4	(Factor, 0.76)		
Zn......... 2.0				
ZnO....... 3.23	Plus, 2.4	(Factor, 0.74)		
SiO_2.......11.6	Minus, 11.6			SiO_2 factor, 1.76
Al_2O_3...... 5.7	Plus, 10.0			Soda factor, 1.60
			0.6 9.1	Fluorspar, 0.1
Fe_2O_3..... 1.3	Plus, 1.5	5.0	Borax factor, 3.79	
			SiO_2 factor, 1.14	
	5.7	5.0	0.6 9.1	

Flux 100 parts of precipitate with 5.7 parts silica, 5 parts borax, 0.6 part fluorspar, and 9.1 parts soda bicarbonate.

When fluxed in this manner the precipitate melts down very nicely. As the charge sinks in the crucible, fresh precipitate, mixed with flux, is added about every 20 min. until the crucible is nearly full of metal and slag in a state of quiet fusion. The slag is then poured into a conical pot, in which a crust is allowed to form to a thickness of 1 in., the molten slag inside the crust being then allowed to run out on cast-iron floor-plates. The crusts from this operation usually contain a small button of gold and silver at the point of the pot, and this button is broken off, and either put back in the crucible, or, if matte be present, is saved for a separate treatment with other similar buttons. The crusts themselves carry sufficient gold and silver to warrant a remelting, while the molten slag that is run on the floor is low in value, and, once a month, is weighed, sampled, and returned to the head of the mill.

After two to three pourings of slag, the molten metals in the crucible are subjected to a refining process, which depends upon the rapid formation of the oxides of zinc and copper when these metals are exposed to the air at high temperature in the molten state. With the furnace tilted forward, the operator throws a handful of bone-ash over the surface of the molten metal. This thickens whatever slag still remains, and enables it to be easily removed by means of spirals of 0.5-in. round iron. The air striking the exposed surface of the metals rapidly oxidizes the zinc and copper. After a couple of minutes, borax is thrown on and dissolves the oxides, forming a slag, which is thickened with bone-ash and removed as before. After about two repetitions of this refining, the precious metals are ready for pouring. The furnace is tilted back and the heat applied for about 5 min. with the cover over the crucible. A step-bottomed truck, with bullion-molds that have been heated and greased with a mixture of oil and flake graphite, is then run in front of the furnace, the mold on the highest step being set in front of the spout. The bars of bullion are poured by tilting the furnace forward over each of the molds in turn. After removal from the molds and cooling, the bars, which weigh from 1,200 to 1,300 oz., are chipped, sampled by drilling top and bottom, and stamped, ready for shipment. By this process, in spite of the high content in copper carried by the precipitate, bars from 850 to 900 fine are obtained.

From 20 to 30 lb. of flue-dust is collected and removed each month from a large chamber outside the melting-room through which the furnace-gases are led. A typical analysis of the flue-dust is: Au, 0.01; Ag, 21.39; Cu, 10.40; Zn, 7.50; SiO_2, 11.08; Al_2O_3, 2.90; Fe_2O_3, 2.30; undetermined, 44.42 (chiefly particles of borax, fluorspar, etc.); total, 100.00 per cent.

For the remelting of slag-crusts, scraps of iron box-bands are added to the slag in the proportion of about 0.5 per cent. of the weight of the slag, and the slag put through the furnace until a button of sufficient size to collect in the mold is obtained. Slag assaying Au, 1.98; Ag, 650 oz. per ton; Cu, 7.5; Zn, 13.5; SiO_2, 28.1; Al_2O_3, 11.34; Fe_2O_3, 4.76 per cent. was reduced in this manner to, Au, 0.06 and Ag, 32.34 oz. per ton; yielding metal that assayed Au, 0.147; Ag, 40.183; and Cu, 42.4 per cent.

Construction-Costs.

The building of the cyanide-plant, the No. 2 mill, and the remodeling of No. 1 mill, were done under somewhat adverse conditions, owing to the revolution in Mexico at the time, and consequent unsatisfactory labor-conditions, but the work was completed in 14 months from the time that the preliminary plans and estimates were approved by the directors of the company.

The following unit-cost figures on the construction-work may be of interest as an example for a property situated in Northern Mexico off the railroad:

Excavation, medium-hard ground with boulders, 16,103 cu. yd. done at an average cost of $0.64 per cu. yd.

Cement masonry, with stone partly dressed, 1,850 cu. yd. done at an average cost of $7.40 per cu. yd.

Cement concrete, 10 per cent. of which was reinforced, 972 cu. yd. done at an average cost of $12.28 per cu. yd.

Carpentry work, buildings, launders and miscellaneous, 320.3 M. board feet of lumber, put in at an average cost (including cost of the lumber) of $84.24 per M.

Steel tanks, assembling, riveting and calking, 247 tons of steel were put up at an average cost of $26.38 per ton.

The total cost of the cyanide-plant was $221,207.79, of which $11,203.39 was spent on testing and preliminary investigations.

The total cost of the No. 2 mill was $50,103.87, made up as follows:

Excavation,	$2,517.68	Supervision,	$1,479.19
Foundations,	9,812.32	Electrical work,	573.23
Carpentry,	14,723.24	Painting,	230.89
Local handling,	761.32	Miscellaneous,	245.86
Machine-work,	2,710.04	First-cost, freight, and duty,	16,990.97
Testing,	59.13		
			$50,103.87

Operating-Costs.

TABLE VII.—*Unit-Costs of Cyanide-Plant Supplies at El Tigre.*

Mine quartz for tube-mills, per lb.,	$0.00379
French flint pebbles, size A, per lb.,	0.02138
Danish flint pebbles, sizes C and D, per lb.,	0.02397
Zinc-dust, 92 per cent. metallic zinc, per lb.,	0.075
Sodium cyanide, 130 per cent. KCN, per lb.,	0.2247
Burned lime, 38 to 40 per cent. CaO, per lb.,	0.0155
Borax glass, per lb.,	0.1375
Sodium bicarbonate, per lb.,	0.07
Fluorspar, per lb.,	0.0745
Sodium nitrate, per lb.,	0.085
Graphite crucibles, No. 125, each,	9.54
Gas-oil for melting-furnace, per gal.,	0.375
Gasolene, per gal.,	0.4385
Hydrochloric acid, commercial, per lb.,	0.119
Lead acetate, per lb.,	0.12

TABLE VIII.—*Consumption of Cyanide-Plant Supplies.*

Sodium cyanide,	5.08 lb. per ton of tailings treated.
Lime, 52 per cent. CaO,	23.0 lb. per ton of tailings treated.
Zinc-dust,	1.3 lb. per ton of tailings treated.
Ratio Zn to Au plus Ag as 1.52 to 1.	
Gas-oil for melting,	0.025 gal. per oz. of bullion produced.
Flint pebbles,	8.7 lb. per ton of tailings treated.

TABLE IX.—*Milling-Costs Per Ton of Ore Milled.*

	November, 1911.		October, 1911.	
	No. 1 Mill.	No. 2 Mill.	No. 1 Mill.	No. 2 Mill.
Crushing	$0.496	$0.413	$0.566	$0.405
Regrinding	0.269	0.021	0.291	0.029
Concentrating	0.961	0.239	1.225	0.521
Construction	0.169	....	0.189	
	$1.895	$0.673	$2.271	$0.955

The cost of cyanide-treatment at El Tigre is very high, owing chiefly to the large consumption of cyanide. The following are representative average costs for the various steps in the process:

Conveying and screening tailings,	$0.11
Tube-milling,	0.65
Classification, elevation, and dewatering,	0.22
Treatment,	1.50
Filtration,	0.30
Precipitation,	0.14
Melting,	0.19
Total,	$3.11 per ton.

Recovery.

From 62 to 64 per cent. of the gold and silver in the mine ore is recovered in the form of concentrates and sorted grade. The recovery by cyanide-treatment of the concentrator-tailings is about 82 per cent., and of the dump-tailings about 85 per cent. The total recovery by sorting, milling, and cyaniding is from 93 to 95 per cent. Owing to the short time that the plant has been in operation, reliable average figures for recovery in the cyanide-plant are hard to obtain, for the only figures that are worth considering are those which are based on production of bullion during periods of at least six months.

Power.

The power for El Tigre is now generated at the power-plant of the Copper Queen Mining Co., in Douglas, Ariz., by two exhaust-steam turbo-generators. At the step-up transformer station in Douglas there are four 400-K. V. A. General Electric Co.'s transformers, any one of which may be held in reserve. The transmission-line, 65 miles long, is designed for 700 kw., capacity 44,000 volts. The wires are No. 4 medium hard-drawn copper, and are spaced 6 ft. center to center in an equilateral triangle. The poles are of wood with treated butts, spaced for an average span of 200 ft. Each pole carries a small cross-arm below the transmission-wires for two No. 10 medium hard-drawn copper telephone wires. The longest span on the line is 1,650 feet.

At the step-down station four 320 K. V. A. transformers lower the tension to 440 volts for operating the various motors in the plant. The transmission-line and transformers were erected under the supervision of Sanderson & Porter, of New York. The efficiency of the transmission is at present about 91 per cent., owing to the amount of power that is used being considerably less than the full capacity of the line and transformers. The average cost of power is now only about one-third of the former cost when gas- and steam-engines at the mine were used.

Interruptions to the transmission from lightning occasionally happen in the summer season during the rains, but seldom exceed 1 hr. in duration.

Acknowledgment.

In conclusion, I wish to acknowledge the many courtesies and the hearty co-operation extended to me by the officers of the Lucky-Tiger Co. during the design and construction of its plant. In particular are my thanks due to James W. Malcolmson, consulting engineer, and to L. R. Budrow, manager, through whose kindness I am enabled to publish much of the operating data which appears in this paper, and from whom I received valuable criticism and advice during the work.

TABLE X.—*The Tigre Mining Co. S. A., Yzabal, Sonora, Mexico.*

ELECTRIC-POWER TESTS IN MILL AND CYANIDE-PLANT, AUGUST AND SEPTEMBER, 1911.

	Electric Horse-Power at Motor by Wattmeter.			
	Starting.	Normally Loaded.	Empty.	P. F.
				Per Cent.
Breaker-Plant.—Capacity 28 tons per hour (35-h-p. Allis-Chalmers motor, 850 r.p.m.). 1 Style K, No. 4, Allis-Chalmers breaker set to 1.5 in.; 1 horizontal 32-in. sorting-belt 28 ft. c.c.; 1 24- by 27-in. revolving-drum ore-feeder; 1 shaking-grizzly 30 by 78 in.; 70 per cent. of ore dropping through the grizzly as fines	12.0	8.4	4.0	
No. 1 Concentrating-Mill.—Capacity 100 tons per day (150-h-p. Allis-Chalmers slip-ring type motor, 1,130 r.p.m., 14 line-shafts in mill, power-transmission from motor).				
Full load except Austin gyratory crusher, 5 vanners, and two Wilfley tables	215	144.7	....	
4 4- by 5-ft. trommels, 1 pair 16- by 36-in. Cole-pattern roughing-rolls. 1 double-compartment bull-jig, 1 bucket-elevator 39 ft. c.c., 18-in. belt, 15-in. buckets	....	32.2	....	
1 pair 16- by 36-in. Cole-pattern roughing-rolls	....	23.4	10.7	
4 double-compartment jigs 23.5- by 35.75-in. also 1 three-compartment jig, 170 $\frac{5}{8}$-in. strokes per minute	....	16.0	....	
1 pair 16- by 32-in. fine rolls	....	18.0	8.0	
1 bucket-elevator 42 ft. c.c., 18-in. belt, 15-in. buckets, speed 360 ft. per minute	....	4.0	....	
2 4- by 5-ft. trommels $\frac{3}{16}$-in. round holes	....	1.2	....	
2 4- by 5-ft. trommels 1.5 mm. round holes	....	0.9	....	
1 5-ft. Huntington mill	....	4.8	....	
1 bucket-elevator 35 ft. c.c., 12-in. belt, 9-in. buckets, speed 360 ft. per minute	....	3.2	....	
1 9- by 10 in. Aldrich triplex return-water pump, 47 r.p.m	....	7.5	....	
10 No. 5 Wilfley tables and 1 shaking-launder	....	10.8	....	
1 10- by 54-in. Frenier sand-pump	....	2.7	....	
Machine-shop line-shafting with shaper and pipe-machine working	....	2.6	....	
11 6-ft. Frue vanners and 2 shaking-launders	....	5.4	....	
Line-shafting driving Wilfley tables and vanners.	....	....	5.3	
Jack-shaft driving Huntington mills with coarse and fine rolls running empty	....	....	26.2	
Jack-shaft driving Huntington mills empty	....	....	3.2	
Line-shafting for crusher, jigs, elevators, and trommels, running empty	....	....	26.8	
Total line-shafting of mill running empty	....	....	48.3	
Ore-Sampling Plant.—Capacity 28 tons per hour, 1 10-h-p. Allis-Chalmers motor 850 r.p.m.				
1 revolving-cone sampler and ore-distributor, 1 Dodge sample-crusher 7 by 10 in., 1 Snyder sampler 27 in., 1 Sturtevant 2- by 6-in. roll-jaw sample-crusher, 1 30- by 78-in. shaking grizzly, $\frac{3}{4}$-in. spacing	....	....	3.48	85
Same running with load on inclined 14-in. conveying-belt assisting the motor	....	3.2	....	85
No. 2 Mill.—(1 75-h-p. Allis-Chalmers slip-ring induction-motor driving stamps and 1 50-h-p. synchronous motor driving tables, pump, and grinding-pan), capacity 100 tons.				
20 1,200-lb. stamps 102 7-in.-drops per minute	....	44.4	....	85
Line-shafting and 12-in. conveying-belt with tripper	....	9.2	....	85
Full load on motor	84	53.6	....	85
Full load on synchronous motor with pump, tables, grinding-pan, and exciter	....	21.4	....	85
1 9- by 10-in. Aldrich return-water pump	....	5.4	....	
1 5-ft. grinding-pan for middlings from Wilfleys,	....	2.9	....	
5 Wilfley No. 6 tables, 4 Deister slime-tables, 4 line-shafts, and exciter motor	....	13.1	....	

TABLE X.—*Continued.*

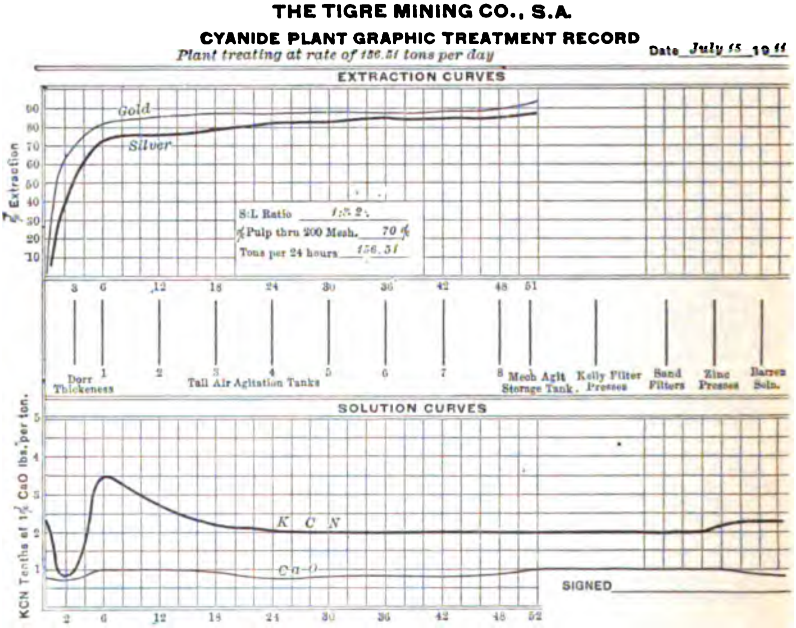
ELECTRIC-POWER TESTS IN MILL AND CYANIDE-PLANT.

	Electric Horse-Power at Motor by Wattmeter.			
	Starting.	Normally Loaded.	Empty.	P. F.
				Per Cent.
Cyanide-Plant.—Capacity 250 tons per day.				
A. Tailings Conveying- and Screening-Plant for Dumps-Tailings. Capacity 75 tons per day				
1 450-ft. 10-in. flexible tailings-conveyor, 1 3- by 5-ft. 6-in. disintegrating-trommel, 1 spiral conveyor, 1 bucket-elevator 25 ft. 5 in. c.c., 8-in. belt, 6-in. buckets, 1 22-in. by 5-ft. 6-in. mixing-trommel, with full load	10.7	5.0		75
Shafting of above without load driven by 10-h-p. Allis-Chalmers motor			1.3	75
Conveying-belt only			2.8	
No. 2 flexible tailings-conveyor driven direct by 5-h-p. Allis-Chalmers motor	10.0	1.6	1.3	75
B. Tube-Mill Section (5 50-h-p. Westinghouse slip-ring motors direct connected to tube-mills, 1 35-h-p. Allis-Chalmers motor driving elevators, classifiers, and dewaterers).				
5 5- by 14-ft wet-grinding tube-mills, each	117	43.5		75
5 42-in. drag-classifiers, 3 Dorr thickeners, 2 bucket-elevators 19 ft c.c., 18-in. belt, 15-in. buckets, speed 400 ft. per min., 2 bucket-elevators same as previous but 40 ft. c.c., 4 line-shafts	16	10.7		75
4 elevators plus line-shaft friction		9.6		
5 classifiers with 3 Dorr thickeners and three line-shafts		1.1		
C. Upper Pump-House (1 25-h-p. Allis-Chalmers motor and 1 line-shaft).				
1 6- by7-in. return-solution Aldrich triplex pump and 1 7- by 9-in. Aldrich triplex return-water pump plus line-shaft friction		14		75
Motor and shafting	3.2		2	
D. Treatment Section.				
1 Sullivan WI 16- by 14-in. duplex belt-driven air-compressor, driven by 100-h-p. Allis-Chalmers motor, compressor making 166 r.p.m. at 30 lb. pressure		90		
1 30- by 12-ft. mechanically-agitated slime-storage-tank and two Tigre slime-thickening tanks with two line-shafts	18.7	6.9		75
30- by 12-ft. agitation-tank only with one line-shaft		5.9		
2 24- by 12-ft. Tigre thickeners and two line-shafts		1.0		
Line-shafting only			0.3	
E. Filter-Plant and Precipitation-Pumps.				
I class "NE–1" Ingersoll-Rand straight-line belt-driven air-compressor, size 8 by 8 in., 210 r.p.m. at 60 lb. average pressure, furnishing 96 cu. ft. free air per min. to Kelly filter-plant, driven by line-shaft from 50-h-p. Allis-Chalmers motor	107.2	15.5	7.5	75
2 6- by 7-in. Aldrich triplex precipitation-pumps.		11.2		
Spiral Ribbon conveyor for discharge of residues from Kelly filters, not in continuous use	34	4	0.7	75
1 20- by 10-ft. mechanically-agitated excess slime-storage tank for filling Kelly filters, with one small pyramid return-water pump for counterweight-tanks	16	2.7	0.5	75
F. Pumps, etc., in occasional use only.				
1 2-in. centrifugal return-water pump	43	20.7	10.7	85
1 1.5-in. centrifugal sand-pump for elevating thickened slime from redwood tanks	20	3.5	0.3	
Miscellaneous Power.				
1 3-h-p. motor for assay-office		2		
1 3-h-p. motor driving ice-plant, capacity 1000 lb. of ice per day	10.7	3.6	1.6	
1 15-h-p. back-geared Westinghouse motor for incline hoist, capacity 8 tons at 200 ft. per minute		18		85
1 10-h-p. Allis-Chalmers motor for carpenter-shop	37	6.7	3.3	
1 14- by 9- by 16-in. WH 2 Sullivan belt-driven air-compressor at 80 lb. pressure making 140 r.p.m		65.5		
at 85 lb. pressure making 140 r.p.m		67.5		
at 90 lb. pressure making 140 r.p.m		69.8		
at 95 lb. pressure making 139 r.p.m		71.8		
Line-shaft with clutch to mine air-compressor thrown out			4.1	

TABLE X.—*Continued.*

ANALYSIS OF POWER-CONSUMPTION DURING AUGUST, 1911.

	Total Monthly El. H-P. Hours.	Average Power when Running.	Total Average Power.
		El. H-P.	El. H-P.
Sorting and Ore-Conveying, including lights	1,283.4	3.45	3.45
No. 1 *Mill.—Stage-Crushing*			167.10
Crushing, including lights	42,360.0	66.50	
Regrinding, including lights	5,431.0	8.50	
Concentrating, including lights	55,437.9	90.10	
No. 2 *Mill.—Gravity Stamps for Crushing*			86.20
Crushing, including lights	37,667.2	62.20	
Regrinding, including lights	5,431.0	3.60	
Concentrating, including lights	9,147.7	20.40	
Cyanide-Plant			401.60
Conveying and screening tailings	2,418.0	7.60	
Tube-milling, including lights	136,743.6	198.30	
Classification, elevation, and dewatering	22,512.6	52.10	
Treatment, including lights	72,108.6	98.70	
Filtration, including lights	17,912.0	25.20	
Precipitation, including lights	8,861.0	13.70	
Melting, lights, and electric drying-oven	3,576.0	6.00	



THE TIGRE MINING CO., S.A.

CYANIDE PLANT GRAPHIC TREATMENT RECORD

Plant treating at rate of 156.51 tons per day

Date July 15 1911

S:L Ratio 1:2.2

% Pulp thru 200 Mesh. 70%

Tons per 24 hours 156.51

SIGNED

FIG. 10.—RECORD OF CYANIDE-TREATMENT.
(Size of original form, 8.5 by 10.5 in.).

The test report and the diagram, Fig. 10, are copied from the forms used during the starting of the plant. The standards given on the test report were predetermined figures based on the results of experimental work, and calculations used in making up the flow-sheet for the cyanide-treatment.

Form 92.

[Size of original form, 8.5 by 10.5 in.]

THE TIGRE MINING CO., S. A.

CYANIDE PLANT TEST REPORT.

Printed figures [here shown in heavy-faced type] denote standards for normal operation on 250 Ton basis.
156.51

Date *July 15, 1911.*

Ton–2,000 lbs.

		PULP.																									SOLUTIONS.			
				Classifiers.																										
				No's. 1 & 2.			No's. 3, 4 & 5.																							
		Overflow Concentrates Bins.	Disch. Settlings From Pond.	Feed.	Sand.	Slime.	Feed.	Sand.	Slime.	Tube Mills, No's. 1 & 2.	Tube Mills, No's. 3, 4 & 5.	Dorr. Thickeners, No's. 1 & 2.	Dorr. Thickener, No. 3.	Wilfley Filter Cake.	Pulp Elevators, No's. 3 & 4.	No. 1. Agitation Tank.	No. 2. Agitation Tank.	No. 3. Agitation Tank.	No. 4. Agitation Tank.	No. 5. Agitation Tank.	No. 6. Agitation Tank.	No. 7. Agitation Tank.	No. 8. Agitation Tank.	Tigre Thickeners, No's. 1 & 2.	Agitation Storage Tank.	Discharged Filter Cakes.	Strong Stock Solutions.	Weak Wash Solution.	Strong Pregnant Solution.	Weak Pregnant Solution.
Siz. Tests.	+ 100 Mesh.			*56.0*	*91.1*					*50.4*	*51.7*																			
	+ 150 Mesh.			*3.2*	*3.7*					*10.4*	*18.3*																			
	+ 200 Mesh.			*2.1*	*2.2*	*9.5*			*36.0*	*4.5*	*11.3*				*39.1*	*28.2*	*26.8*	*28.2*	*31.8*	*30.2*	*28.2*	*29.4*	*29.7*							
	— 200 Mesh.			*38.7*	*3.0*	*90.5* **90.**			*64.0* **90.**	*34.7*	*18.0*	**90.**	**90.**		*60.9* **90.**	*71.8*	*73.2*	*71.8*	*68.2*	*69.8*	*71.8*	*70.6*	*70.3*							
	Sp. Gr.			*1.105*	*1.50* **1.48**	*1.04* **1.04**		*1.45* **1.48**	*1.14* **1.11**	*1.50*	*1.45*	*1.45* **1.43**	*1.22* **1.21**	×	*1.24* **1.25**	*1.16*	*1.10*	*1.18*	*1.18*	*1.20*	*1.20*	*1.20*	*1.20*	No. *1 & 2* *1=1.38* *2=1.20*	*1.31* **1.32**					
	S. to L. Ratio.			*1:6* **1:6**	*1:1–*	*1:17* **1:17**		*1:1+*	*1:5-* **1:5**			*1:1–* **1:1**	*1:2.3* **1:2.4**		*1:2.2* **1:2**	*1:4*	*1:3.5*	*1:3.5*	*1:3.5*	*1:3.2*	*1:3.2*	*1:3.2*	*1:3.2*		*1:1.6* **1:1.5**					
	Per Cent. Moisture.				**50.**			**50.**				**50.**		**30.**												**40.**				
	Per Cent. K. CN.														*.080*	*.35* **0.35**	*.275*	*.210*	*.190*	*.200*	*.205*	*.205*	*.210*		*.210*		*0.23* **0.26**	*0.16* **0.05**	*0.20* **0.24**	*0.16* **0.05**
	Ca. O. pounds.														*.90*	*1.05* **2.4**	*1.00*	*.95*	*.75*	*.80*	*.80*	*.80*	*.85*		*1.20*		**8.**		*1.00*	*.85*
Assays.	Gold. oz.	*Average of all Tailings for day.*		*.079*							*.03*	*.05*	*.018*			*.015*	*.01*	*.01*	*.01*		*.01*	*.008*	*.008*		*.006* **.005**	*.005*	*.004* **.005**	*.001* tr.	*.024*	*.014*
	Silver. oz.			*17.22*							*10.47*	*20.95*	*5.58*			*4.69*	*4.09*	*3.59*	*2.99*		*2.69*	*2.69*	*2.69*		*2.39* **.70**	*2.47*	*0.42* **.30**	*0.21* **.08**	*5.15*	*3.05*
	Values U. S. Cy.			*$10.71*																					*$1.39* **.45**	*$1.41*	*$0.24* **.25**	*$0.13* **.04**	*$3.06*	*$1.81*
Pr. Ct. Ex.	Gold.										*62.*					*81.2* **.57**	*87.5* **.78**	*89.5* **87.**	*89.5* **91.**	**92.**	*89.5* **92.5**	*89.8* **93.5**	*89.8* **94.**		*92.4*					
	Silver.										*39.2*					*72.7* **41.**	*76.3* **65.**	*79.3* **77.**	*82.5* **83.5**	**86.**	*84.4* **87.5**	*84.4* **89.**	*84.4* **90.**		*86.2*					
	Values. U. S. Cy.															**43.5**	**67.**	**79.**	**85.**	**87.5**	**89.**	**90.**	**90.5**		*87.0* Pr.Ct.	*86.8* Pr.Ct.				
Av. Rate Fl.	Gals. per min.			**192**								**19**	**110**		**125**												**83**		**83**	**125**
	Total Tons per 24 hrs.			**1148**								**112**	**659**		**750**												*338* **500**		*338* **500**	*660* **750**
	Dry Tons per 24 hrs.			**164**			**340**			**164**	**340**	**56**	**194**	**56**	**250**															

Total Tailings Treated for day —156.51

Signed________________________
Cyanide Foreman.

[TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS.]

Alaska Coal-Land Problems.

BY H. FOSTER BAIN, SAN FRANCISCO, CAL.

(San Francisco Meeting, October, 1911.)

[SECRETARY'S NOTE.—This paper, presented in oral abstract at the San Francisco meeting, was not at first supposed by Mr. Bain to be required for publication in the *Transactions;* and the excursion to Japan, in which he took part, immediately after the San Francisco meeting, delayed his presentation of the manuscript. After his return to San Francisco, Mr. Bain, in view of the importance of his paper and its discussion, kindly furnished it in full. The delay in its publication (since it was received in April last) has been due, partly to the circumstance that other contributions had meanwhile acquired the "right of way," and partly to the consideration that, the situation contemplated in Mr. Bain's paper having remained practically unchanged, the paper and its discussion would not lose force or timeliness by this lapse of time.—J. S.]

No acute observer of public affairs can have failed to observe a growing disposition on the part of the public to change emphasis in government from political equality to social justice. When our own government was founded it was felt to be sufficient to secure to each individual the fullest possible freedom of opportunity, and such a scheme was excellently adapted to the rapid development of a large territory having a small population. As the number of people in a given area increases, there are more points of contact and conflict; each must move in a smaller orbit, and there must be more regulation. Germany is making faster progress in industry than is any other country. The principal reason is the insistence there on team-play. In Germany there are fewer multimillionaires and more socialists (spelled with either a big or little "S") than in any other of the leading countries; but Germany wins trade and is steadily improving the conditions of life for her citizens. To some, the change from the freest individualism is welcome; to others it is intensely unacceptable. To this fundamental difference in view-point may be imputed much of the vigor with which the controversy regarding coal-lands has been waged. That present conditions in Alaskan coal-fields are extremely unsatisfactory is generally recognized, but there are wide differences of opinion as to the proper remedies. Large areas of excellent coal within a few miles of tidewater remain unopened, while railroads and other enterprises import coal at a cost of from $7 to $9 per ton on the dock. In building and operating the Copper River & Northwestern railroad to June 30, 1911, British Columbia coal to a total value of $509,943, and Washington coal to the value of $49,438, was imported. The railroad and the leading coast industries have now been converted to burn oil imported from California. In the interior, the price of coal is practically prohibitory, and development of any industry requiring handling of large tonnages is impossible except at the seacoast. Industry in Alaska always will be at a disadvantage, owing to the difficulties and expense of navigation along its coasts, and of transportation into the interior. This, however, is all the more reason why full advantage should be taken of any favorable local feature, such as abundant and excellent coal.

In examining the matter of the opening of the Alaska coal-fields, two distinct questions must be considered. The first, and more pressing, is how to get coal on the market quickly in quantities sufficient to satisfy immediate demands. The second is, what disposition shall ultimately be made of the coal-lands, and under what conditions coal shall be mined in the future. Both questions involve railway problems and are complicated by existing rights of coal-land locators. It is not likely that any satisfactory solution of the second problem will be found immediately. The ramifications of the matter reach such varied interests, and so many things must be taken into account, that a quick determination might well prove a wrong determination. The first problem, however, that of promptly furnishing Alaskan coal in such quantity as is really needed, seems capable of speedy and safe solution.

It has been estimated by A. H. Brooks that there is a present potential market for 1,000,000 tons of Alaska coal per year. Whether or not these figures be correct, it is undoubtedly true that there is an important market awaiting the opening of the mines. At present all the high-grade coal used on the Pacific coast comes from the East or is imported. The navy, for strategical reasons, uses only the best grade of Pocahontas coal; and when the fleet goes to sea it is followed or preceded by a long string of colliers carrying Eastern coal. At all times there is a procession of boats traveling around South America in order that the battle-ships and cruisers shall have smokeless coal. Necessarily, much coal is burned in bringing this fuel to the ships, and, necessarily, also, the cost of the coal delivered is greatly above that at Eastern tidewater ports, where it has already been raised by the several hundred miles of railway-haul from the mines. In case of war, the supplying of ships in the Pacific with Eastern fuel would be extremely difficult and expensive. Even now the navy is dependent upon foreign tramp steamers. So far it has not seemed wise to convert the whole fleet into oil-burning vessels, because of the difficulty of buying oil in foreign ports. In the opinion of experts, the Bering River and the Matanuska coal-fields of Alaska can furnish a grade of coal excellently adapted to naval use, and the opening of mines there is important in order to afford an adequate and at all times accessible source of supply. The navy uses ordinarily about 150,000 tons of coal per year on the Pacific coast. Possibly the other departments of the government use as much more. In addition, there is a local market in Alaska, now supplied from British Columbia and Washington at considerable expense. Beyond is a strongly competitive market, in which the dominating features are the abundance, low price, and high heating-value of California crude oil. In the three Pacific States together, in 1910, imports and local production of coal amounted to 4,600,000 tons. E. W. Parker has estimated that in the same year fuel-oil displaced 12,000,000 tons of coal in these States. To what extent Alaska coal can enter this competitive market is uncertain, but the non-competitive market is not unimportant. In it, owing to naval requirements, the government is the largest factor. The obvious thing to do would seem to be for the government to open its own mine and supply its own coal, at the same time meeting the local demands of the non-competitive market by selling the surplus coal at a small advance on the cost of production, as has been suggested by Walter L. Fisher, Secretary of the Interior. The opening of a mine by the government in no way prejudices any future settlement of the larger problem of final disposition of the lands. It will, however, remove any fancied necessity for a hurried decision of the main matter, will relieve the local fuel-situation, will lead to just as much development, and the employment of just as many men, as if some private syndicate undertook the work, and will permit the government finally to act on the larger questions involved with exact knowledge based upon actual experience under the conditions to be met. It has long been customary for large coal-users to maintain a coal-mine, or to have an interest in a coal mining company, so as to have full information regarding costs when making their yearly contracts. For years the Chicago, Milwaukee & St. Paul railroad ran a small coal-mine on this basis, and the information it obtained was worth many times the risk of loss. Why should not the government safeguard the people's interest as carefully as a board of directors for a company looks after those of the stockholders?

Various suggestions have already been made as to how the government should proceed in case it be decided to open a government mine in Alaska. An independent commission has been proposed, but this is, I think, unnecessary and likely to prove expensive. It has also been suggested that the army be placed in charge, and the merit of this lies in the fact that the people have entire confidence in the non-political character of the army and the honesty of its officers. The excellent work, in particular, done at Panama, has greatly raised the popular respect for the Engineer Corps of the army. It is also true that to meet a similar scarcity of coal in the Philippines, the army opened a colliery there and conducted it not only ably, but in such a manner as to demonstrate the value of the coal and of the field, and this led ultimately to the opening of private mines. The army, however, has other work than mining coal, and no good purpose is served by diverting its officers from their own sphere. Aside from that, the government has in the Bureau of Mines a corps of men especially well qualified to handle this particular matter. In this bureau are all the experienced men necessary to man a colliery, from pit-boss to general manager. Attracted by the opportunity for travel and study, an unusually capable and well-trained lot of engineers has joined the service. A better manager for the proposed enterprise probably could not be found than the chief mining engineer for the Bureau, who has seen entensive service in the coal-fields of Colorado, Iowa, and Illinois, and has served as engineer, superintendent, general superintendent, and consulting engineer for a number of large companies. He has opened, equipped, and run a number of collieries as large or larger than any needed in Alaska, and is thoroughly familiar with coal-mining, not only in every part of the United States, but in England, Germany, and France. There are other men in the Bureau who have had valuable if not as extensive experience, and they are all men of high character, of just the sort to place in charge of a difficult enterprise. Put the matter in their hands with sufficient capital and adequate authority, and the mine will be well and honestly run. Incidentally, it will furnish the Bureau an excellent opportunity to test each new discovery made in its laboratories, and to try out, on a working scale under commercial conditions, its various recommendations for increasing safety and decreasing waste in mining. To the objection that will promptly be made—the high cost of government work—it may be urged that the facts by no means point conclusively to a necessarily higher cost. The Reclamation Service has repeatedly built on force-account at prices below bids from contractors, and the great Los Angeles aqueduct is being built by the City Engineers, and well built, at a cost quite as low as any firm or corporation has ever attained on similar work. On one division the work was done for approximately $2,000,000 less than the lowest bid. Incidentally the Reclamation Service itself has opened and run a government coal-mine, and done it well. Under these conditions it is hardly worth while to answer the old statement that "the government can't run a mine." It can and does. Get good men and give them reasonable support, and they will build as cheaply for the government as for a corporation. The common notion to the contrary, is based on the excessive cost of jobs let to favored political contractors.

In opening a government coal-mine in Alaska, it is important that conditions be made as nearly as possible identical with those under which a private company would have to work; otherwise one of the large purposes of the enterprise, the obtaining of exact information regarding mining-costs and other conditions, will be defeated. A definite amount of land should be reserved by the Secretary of the Interior, and allotted to the project. Since 2,560 acres is the amount permitted one company under the law of 1908, the same acreage might well be set aside for the government mine. If it is too small for the government, it is likewise too small for a private company, and if large enough for one it is sufficiently large for the other. A definite amount of capital should be allotted to the project, and the coal furnished to the navy and other departments should be credited to the enterprise at the same rate as when sold to individuals. In figuring costs depreciation should be fully taken into account, since the plant will need to be renewed from time to time. A sinking-fund should also be accumulated, and in every particular the enterprise should be run on a business basis.

Since the coal is valueless unless brought to the coast, transportation-problems must enter largely into any solution of the coal-problem. This is true whether the government or some company mines the coal. To reach the Matanuska field, which is favored by Mr. Fisher, it will be necessary to take over and extend the Alaska Central railroad. As this is in the hands of a receiver, its owners would probably welcome the opportunity to sell to the government. The most difficult part of the road is already built, though much money would be needed to put it into good condition. Owing to grades and curves between Seward and Knik, the line would not be easily or economically operated; and large tonnages over this division would be expensive to handle. It is proposed to establish a summer harbor at Knik, and to ship the bulk of coal from this point, to which railway-transportation is easy. The line to Seward could handle quick freight and afford an emergency-route for coal. Mr. Fisher has recently suggested that the men and equipment soon to be out of commission at Panama be transferred to Alaska, and employed in building the proposed trunk-line from Seward to Fairbanks. There is much to commend this suggestion, since, under the plan proposed, a tried and effective force would be made available for the work.

As to the need of a trunk-line to the interior of Alaska there can be no question, if the country is ever to be developed. The practical question is wholly as to means and method. Into the Bering River coal-field, the one most easily accessible, a number of surveys have been made. There are, however, only two routes that need to be considered. The first is a direct line approximately 25 miles long to Controller bay or Katalla. This requires the making of a harbor, and any estimate of expense of shipment by this route must take this into account. Eventually one or more such lines is likely to be built, but for the present the cheapest route would seem to be the longer one to Cordova. This requires a road from the mines almost to Katalla, then north to a junction with the Copper River & Northwestern railroad. The distance is longer, 58 miles, but of the total, 33 miles form part of the main line of the road already built, and the cost of an extra harbor is avoided. This is the more feasible route for the present. It would have the advantage of permitting through shipment to the copper-mines of the interior, while the other would require transfer to boats at Katalla and retransfer to cars at Cordova. A considerable part of the local benefit to be derived from opening the mines would be lost if connection were not made with the Copper River & Northwestern. The tonnage that would be furnished by the government's own coal-requirements would probably be sufficiently attractive to cause one or the other of these roads to be built by private capital in case the decision favors the Bering River field. Wherever the work be started, the Secretary of the Interior or the Director of the Bureau of Mines should be authorized to negotiate for transportation to the sea on a basis of an agreed annual tonnage determined by naval requirements, and fixed rates reviewed by the Interstate Commerce Commission. If, on these terms, private capital be not forthcoming for a railway line, the latter should be built by the government and operated directly or through lease, as might be determined at the time. The cost of the road to the coal-fields would be less than that of colliers to bring coal to the navy from the East, and the cost of operating the whole mining enterprise, granted that the worst estimates of the excess-cost of government work prove true, would be less than the excess-price now paid on the Pacific coast for Eastern coal. It is taken for granted that arrangements would be made for the navy and other government departments to use the government coal on the Pacific coast. It would probably be necessary to establish supply-stations at the tidewater terminals, on Puget sound, and perhaps elsewhere, and to deliver coal to them either by contract or by using government colliers. If any coal be sold outside Alaska (and as to the advisability as well as possibility of that there would be much question), the cost of transportation should of course be added to that of mining.

Whatever be the decision as to government ownership and operation, leasing, or any other method of opening the Alaskan fields, existing rights should be adjudicated as promptly as possible. Under the terms of the Cunningham decision, if they be adhered to, it seems doubtful whether many, if any, of the claims will ever go to patent. In any event, final decision cannot be expected promptly; and what is needed now is the prompt opening of one or more mines. If any of the existing claims are valid they should be promptly passed, since the opening of a private mine or two would supplement rather than conflict with the government colliery, and would permit the latter to be devoted practically to furnishing coal for the navy and other government departments. The potential competition of the government mine would prevent any unreasonable increase in local prices, even in the event that actual competition of oil did not operate to do so.

The adoption of the plan outlined would (1) make the navy independent as to fuel-supply in the Pacific; (2) furnish coal at reasonable cost to local industries needing it; (3) permit the government to determine the exact facts as to costs and conditions of mining; (4) give the Bureau of Mines an excellent field for testing its theories and recommendations; (5) permit adequate time to be taken to adjudicate existing claims, and formulate a really satisfactory policy of handling the coal-lands of Alaska. Adoption of the plan outlined would not dispose of the main question, the final disposition of Alaskan coal-lands. I am by no means sure that a satisfactory final solution can now be formulated; and until it is clear that the right method has been found, I, personally, would favor leasing whatever acreage may be needed for the fullest development, upon terms outlined later. It should be remembered that the United States is the owner of these and other public lands as trustee for the whole people, and Congress is the sole judge of the terms under which they may be occupied or sold. Whatever, in the judgment of Congress, is for the greatest benefit of the people of the United States, not Alaska alone or the Pacific coast, is the right and proper thing to enact into law, and what has been wisest in the past may very well not be, in the judgment of Congress, now equally wise.

In discussing the Alaska coal-land situation it is commonly assumed that the matter is primarily one of conservation. This is far from being true. Any analysis of the speeches that have been made, and of the papers written on the subject, will make clear that distribution of profit rather than waste of coal is what the proponents of change have chiefly in mind. Engineers, on the other hand, are primarily concerned with development, and as conservationists their interest centers mainly upon securing development with the minimum of waste. As has been repeatedly pointed out, development can be expected only if some profit accrue to, or be anticipated by, the developers. Avoidance of waste is a matter of securing the best technical service in the actual conduct of an industry. Economic considerations determine the conditions under which development may or must proceed; the engineer determines how best to meet a given set of conditions. Any circumstance which permits more money to be spent on a given process or a larger control of an industry to be exercised, makes possible a greater saving of material. Higher prices and larger units of business organization make for a less waste of coal or any other material. A single syndicate mining and selling Alaska coal would undoubtedly conserve both coal and capital better than any other organization. To turn the matter over to such a syndicate would be the simple and business-like method of procedure—if these were the only considerations. It is evident, however, that such a bald solution of the problem would satisfy no one except the members of the syndicate, and some other method or some modification of this one must be found if progress is to be made.

Recurring to the statement, then, that as a people we are moving away from our early intense individualism, it is clear that there is now a widespread desire to protect the average citizen from extortion following monopoly, and to prevent the accruing to individuals of the unearned increment on mineral lands. The problem, as thus stated, is world-wide and it is to be faced in every industry. Its solution as regards the existing public mineral lands would be but an incomplete answer to the great needs of present civilization—(1) the realization of the benefits of large business organizations without loss of control of monopoly; (2) the equitable distribution of the surplus earnings of mankind. Solution of the Alaskan coal-problem will be at best but one small step in the solution of the general problem, and any method applied in Alaska and based upon government ownership of the land can be of but small immediate importance, since for many years to come the people of the United States will draw the bulk of their fuel from fields that have already passed into private ownership. It is true that approximately one-third of the known coal is still owned by the general government. This, however, is undeveloped. The practical question is, whether it can be opened under regulations that will rectify mistakes flowing from past policies, or at least prevent similar mistakes in the future.

The great anthracite-deposits of Pennsylvania are already in the hands of a monopoly, and in a large measure the enhanced value of the land has already gone to the individuals concerned in organizing and maintaining it. There is no question but that the industry has been benefited, but there is widespread popular dissatisfaction with the control exercised, in this case by private individuals, and the government finds itself estopped from effective regulation of the industry by the fact that the lands are owned privately. Recent efforts of the government to enforce the Sherman Act, while nominally successful, seem to have proved barren of actual result. In the bituminous coal-mining industry, on the other hand, where intense competition is the rule, conditions are chaotic and intensely unsatisfactory to every one except those large users of steam coal who put low price above every other consideration. The average citizen who buys in small quantities is already paying high prices. By stringent anti-monopoly laws, the tendency towards monopoly was halted before the bituminous coal-mining industry became completely organized. It would, however, be extremely difficult to reach a fair and disinterested conclusion as to whether, from the point of view of the general public, present conditions are more unsatisfactory in the anthracite or the bituminous fields. Low prices do not tell the whole tale. Wasted capital, wasted coal, low wages, and lost lives, the formation of a large class of permanently discontented workmen—all these must be counted against low prices of coal used by railroads and power companies, granted even that low prices to the latter mean some benefit to the ultimate consumer of transportation and power.

An instructive lesson may be drawn from the anthracite-fields. Many persons forget that for years coal was mined and sold there under fiercely competitive conditions. At a time when it cost but $1,000 to open a mine, and $10,000 sufficed "for quite a respectable business," many rushed into the industry. Overproduction worked the inevitable result of low prices and no profit. Since, however, there was a profit to the railroads in transporting the coal to the market, mining was stimulated by loans from the railroads to the struggling coal companies. Eventually the railroads had to take the mines to protect their loans, and then discrimination in rates and service became inevitable. In time the cost of opening mines rose to almost prohibitive figures, and no one could afford to invest the large sum necessary unless he also controlled transportation. All these factors worked together to produce the inevitable result, a monopoly of anthracite-mining in the hands of the railroads. From many points of view the results have been excellent. The properties, however, are operated, not for the sake of the mines, but for the railroads, and a great national industry is subordinated to the immediate necessities of a group of roads. For many years there was no profit in the actual mining of anthracite; probably there is little now. Millions have been paid in royalties to fortunate owners, more millions have been made in the business of transporting and marketing the coal; but in the business of mining there has been no considerable profit. There has always been a surplus of mines and of railroads in the anthracite-fields, and, the larger capital being invested in the railroads, it was inevitable that they came to dominate the situation. The State and the National legislators never aided the industry by constructive legislation based upon any real consideration of the larger problems of the people and the coal. When they acted they merely pecked at the industry with petty restrictive acts. The net result is a firmly-seated private monopoly of a great national resource.

In the bituminous coal-fields the story is similar to that of the anthracite-fields in the early years—an abundance of coal and of capital, investments made on the basis of an inflated industry during "boom" times, overproduction, idle plants and men, wasted money and coal, attempts of the railroads to control production, each in its own interest, and the formation of monopolies in limited areas. The tendency towards a larger monopoly has been checked by anti-monopoly legislation and the regulation of interstate commerce. In the meantime many idle colliery-plants stand gaunt and rusting in the valleys and on the prairies, and an idle plant does no service for any man. It is useful only as a club in considerations of rates and prices. Clubs, it is well to remember, are the weapons of the stone age.

The two industries exemplify the effects of combination on the one hand, and of competition on the other. In 1891, under competitive conditions, anthracite sold at the mine for an average price of $1.73 per ton. In 1910, under monopoly conditions, the price had risen to $2 per ton, an increase of 16 per cent. In the same years bituminous coal, under competitive conditions, rose in price from $0.99 to $1.12 per ton, an increase of 13 per cent. These figures are no measure of retail prices, but it is evident that whatever may have been the effect as regards consumers, the creation of a monopoly in the anthracite-fields has benefited producers of coal more in other directions than by increase in price. It is important to remember in this connection that the cost of selling and delivery is a large element in the retail price of coal, as of other products, and often the buyer of small lots imputes to the original producer charges that originate long after the coal reaches the top of the ground.

It seems clear, on the whole, when the two industries are contrasted, that combination rather than competition yields better results. To secure its full benefits without injury to the public is the real problem, and some form of public control is evidently desirable. The simplest form of control yet found is ownership. It does not follow that public ownership of all the coal-mines is either desirable or necessary, but, so long as the government already owns one-third of the coal-reserve of the nation, this coal should not be allowed to pass into private hands until some adequate means of meeting the problems outlined has been found and tested. It has repeatedly been proved that ownership of a majority of the stock is not necessary in order to control a corporation, and a holding of one-third of the coal is so much greater than any possible combination of lands privately owned, that, particularly in combination with its other powers, the government can determine absolutely the future of the coal-business of this country. The government ought not to sell the coal-lands until the whole matter is settled. In the meantime, all the demands for development can be amply met by a simple leasing-system, such as has been proposed by the administration. There is no benefit to be derived from locking up the coal-lands in either government or private hands, but to sell them irrevocably, at nominal prices and in quantities in excess of the needs of present development, is on a par with the giving of perpetual franchise for municipal utilities. The leasing-system may or may not prove the correct procedure as a permanent policy, but for the present it has many advantages over any other.

The problems that now confront our bituminous coal-mining industry arose a few years ago in the same form in Germany. In that land of orderly industry, the coal business was in as bad condition as was ever that of Pennsylvania or any other American State. There was this difference: the Prussian government, as the owner of many miles of railroad, was concerned, in place of, as in America, a group of private individuals; but, further, the government there did not, as here, own a large part of the known coal. Difficulties in railroad-operation, as well as the general disorganization of industry, focused public attention on the coal industry. The State railways sometimes enjoyed cheap coal, and at other times paid dearly for it. At times the lines had much traffic, and at other times little. For short periods the collieries made large profits, and these periods were succeeded by long lean years. The methodical Germans studied the situation, and then met it in a practical manner, undeterred by theoretical considerations of what might happen. The government purchased enough mines to give it control of approximately 40 per cent. of the coal-producing capacity of the country. Then, on the basis of the ownership of these mines and a thorough knowledge of the business, the government joined in a syndicate for mining and marketing coal. While there has been some criticism of the results, there has been steady work, moderate but regular profits, and better prices. English coal companies, it is true, have made inroads on the Berlin market, but the Germans gaze with equanimity upon this. They regard the low-priced coal furnished by competing English companies as just so much gain. Germany has regulated the coal-monopoly by becoming a party to it.

I am among those who believe that "trusts," meaning thereby large business organizations, are inevitable. The conditions of modern industry favor them, and their economic advantages outweigh their evils. It is not necessary, however, to allow them to proceed unregulated, and to do so is as foolish as to allow unrestricted employment of any other new device, such as automobiles or wireless telegraphy. Trusts need regulation, but if business is to be conserved, it must be by informed and sympathetic regulation. The best way to control a trust is through the board of directors. Even a minority there will be listened to with respect, and suggestions, if sensible, will be adopted.

In Alaska the conditions of industry are naturally adverse. It is extremely difficult to successfully develop an enterprise, and only the most efficient form of organization can hope for more than sporadic success. The coal-fields may be developed more quickly, economically, and efficiently by a single great syndicate than by a number of small warring concerns. The first step in the solution of the Alaskan coal-land problems should be the opening of a government mine, primarily to supply the government needs of coal. If necessary, the government should build the needed railway to the mine. The second step should be the opening of private collieries on leased ground, and the formation of a selling syndicate to pro-rate all contracts. The government itself should be a member of this syndicate and should be represented in its management by an expert resident official of ample authority. If any of the present claimants of lands succeed in getting patents, they should be invited to become participants in the syndicate, and they would probably find it greatly to their advantage to join.

The immediate thing is to provide for mining some coal now and still leave the policy of the future to be shaped to meet its own needs. This seems to me to be most easily accomplished by following the opening of the government mine with an offer of a limited number of leases to any one who may care to undertake the risks involved in opening collieries in Alaska. These risks, from a business point of view, are bound to be large for many years, and the government in offering a lease should be prepared to safe-guard in every reasonable way the lessees as well as the public.

Leases should be granted upon liberal terms, fixed in advance, should run for a period of preferably 25 years, and should cover in each case sufficient area to permit continuous operation of a modern plant through the life of the lease. The Mining and Metallurgical Society has suggested a lease for all the coal in the ground, with a minimum required production that rapidly increases with acreage, in order to afford security and yet not permit holding ground for speculation. There are many possible ways of meeting this problem, and any one of several will prove satisfactory, provided it be clearly announced in advance and remains unchanged during the operations of the lease. The royalty should be low, since a healthy industry is more important than revenue to the government, and it is no proper function of the government to make money out of one industry or class of citizens to the profit of others. The number of leases operated at any one time should be restricted to correspond to the probable market for coal. Whenever, however, the existing collieries become unable to supply the demand, or the profit on the business for two successive seasons amounts to more than a fixed percentage on the capital invested, due allowance having been made for maintenance and sinking-fund, the government should offer additional blocks of ground for lease, and the new collieries, when open, should become members of the selling syndicate and should receive their *pro rata* of contracts. Leases had best be let by competitive bidding. By this method the companies would fix their own royalties. In the initial competition they should be allowed to offer alternative bids based upon the number of leases that it may be decided to let. Clearly, a company having the whole business can pay a higher royalty than one which must divide the trade with others. It is true that competitive bidding has disadvantages as compared with assignment by lot or in order of application, but among those familiar with the business and having the capital to carry out the terms of the lease, bidding would be equal, and there can be no advantage in having others undertake to open collieries.

All applications for leases should be accompanied by bond, and after two years within which to open the mine, royalties should be collected for the proportionate tonnage of each year whether coal be mined or not. Such royalties should apply on future production, their payment being merely required to prevent indefinite suspension of operation. The detailed terms of the suggested leases might profitably be varied somewhat, but the following general principles should be held to: (1) Terms liberal and known in advance; (2) all companies to participate in sales and contracts; (3) the government to be fully informed of all details; (4) the government to be in position to prevent extortion by permitting the opening of additional mines; (5) royalties to be determined by the companies themselves by means of competitive bidding; (6) revenue over and above the cost of maintaining the service should go in large part, if not entirely, to the local Territorial government; (7) leases should be freely transferable, and subject to cancellation only for material failure to observe their terms and after court review.

Until the California oil-fields begin to show signs of depletion, it is not likely that development of an Alaska coal-mining industry will be rapid, but under the plan suggested above, every opportunity would be afforded for opening mines as fast as the market could be developed, and that without sacrifice of any interest, present or future. Such a complete change in policy should not be attempted on too large a scale at once; and for that reason it is suggested that the plan be applied to Alaska only at present. If the results prove satisfactory the same system can be extended to the remaining coal-lands in the States, and there can be little question that a selling syndicate of this sort could bring to the bituminous coal industry all the benefits now apparent in the anthracite-region, without sacrifice of public interests. The advantage of the leasing-system is that it does not unduly commit the future of the industry, and as new leases came to be given from time to time the companies would offer higher royalties if the business warranted, or the price of coal would fall. In either event the "unearned increment" would accrue to the public rather than to private individuals. The plan does away with speculation in the land, which is exactly what it is designed to accomplish. It limits the amount of speculation possible in leases by fixing a maximum profit that may be realized. By way of compensation it also limits the chances of loss by restricting competition, and by throwing on the government much of the expense of pioneering, in that the government mine, to be opened first, must have transportation-facilities, and later comers may enjoy these equally with the government. It meets or attempts to meet the Western objection to leases, namely, government from Washington, and constant interference as to details, by requiring the terms of the lease to be fixed in advance and that leases shall only be canceled in the same manner and on the same grounds as are private contracts. A further important feature of the plan is a resident official with ample authority to exercise the somewhat limited discretion proposed. No attempt is made to fix prices. These will be determined by competition of Canadian and American coal, California oil, and the cost of hydro-electric power. The limitation is as to profits, and that is to be enforced in the normal method by bringing in additional competition when profits become too large. An orderly rather than speculative method of setting this force into play is provided. The official whose duty it will be to offer new leases will be fully informed as to the facts and must proceed according to fixed conditions known in advance. Finally, his judgment will be put to test, since, unless the showing of large profits or insufficient supply be satisfactory, no competent bidder will be found when the leases are offered. In the meantime the producers are protected against cut-throat competition among themselves, the power of the government is available to compel fair transportation-rates, and the selling of the product, the most difficult and delicate part of the coal business, is in the hands of the companies and not of the government. The plan provides for placing the producing companies upon the best economic basis that competition of other fields will permit, and under these conditions they will be prepared to mine coal according to the best practice that is now possible. Conservation of coal and protection of life in the mines must be enforced through the police power of the State and regulations directed to that end must be matters of law.

In order to secure good regulations and their vigorous enforcement, however, the operating companies must be placed in a satisfactory financial position. This is only possible when they are protected against unfair competition and unreasonable capitalization.

It is too much to assume that any plan will prove entirely satisfactory, but the above is offered as one which at least takes account of the difficulties and provides means of meeting those that can be foreseen.

DISCUSSION.

R. W. RAYMOND, New York, N. Y.: Mr. Bain's introductory statement that there is a growing disposition on the part of the public "to change emphasis" in government from "political equality" to "social justice" is somewhat vague; but I gather from his argument that he approves the growing disposition to which he refers, and that the "emphasis" which it demands involves a radical change in the policy hitherto pursued by the United States—the policy under which the Pacific States and Territories have been developed thus far. I do not understand how "social justice" is to be furthered by the sacrifice of "political equality," or how the change is to be effected by "emphasis." But I fancy that the beneficent and necessary new system of government which Mr. Bain has in mind is the administration of the natural resources of the public domain for the profit of the whole nation (*i. e.*, of the Federal treasury), instead of transferring them to individual citizens, or to the States and Territories in which they are situated, to be utilized as a source of growth and prosperity by those communities.

The general discussion of this proposed reform I have undertaken in a separate paper (announced for the San Francisco meeting, but withdrawn when Mr. Bain's paper was announced), which I hope to present at the next opportunity. In that paper, I shall argue that, in the light of the history of our public lands and our land-laws, a change of our policy at this time would be a handicap upon the progress of certain States and Territories, and would ultimately result in a Union of States, some of which, by reason of events in their history, could not exercise, as to their own natural resources, the powers secured to others. Another part of my historical argument consists in the narration of an experiment in the Federal administration of public mineral lands, undertaken under conditions much more favorable than would surround such a venture now, pursued for 40 years, and finally abandoned in despair, by the vote of an overwhelming majority of Congress, acting upon the recommendations of a Presidential message. In these days, when "emphasis" seems to have superseded even memory, that story may surprise many who think they are now proposing a new substitute for the old system which our fathers established, but which, after serving us well for a century or more, must now be pronounced inadequate—with due apology to the said fathers, who "did not foresee," etc. It is not fashionable to recall anything prior to the war of '61; some ardent reformers even seem to think that history, as a useful guide, begins with the close of the Spanish war, when the United States, having picked up the island of Porto Rico, the island of Guam, and the 1,500 islands of the Philippine archipelago, became a "world-power." To such I can recommend the story, which will at least show them that the plan which they regard as necessitated by new conditions, and suggested by a new generation of progressive statesmen, is simply an old failure, revived by ignorant enthusiasm. It is not "progress," but reversion to an obsolete ancestral type.

Another proposition defended in my paper is that, if governmental interference with private enterprise is warranted at all, it should be confined, so far as possible, to local governments. Apart from all questions of constitutionality, it is the verdict of a thousand years of experience that the government which regulates the affairs nearest to the citizen should be itself nearest to him. This I shall support in my forthcoming paper with instances from our own history and that of other nations; and I shall here offer, under this head, but a single remark.

Namely, mistakes in legislation, or abuses in administration, are much more easily exposed and remedied, if committed by the government of a single State or municipality, than if they occur as features of a national policy, and its enforcement by Federal Executive Departments. The wrongs suffered by the citizens of a given State, for instance, can be removed by arousing the sense of justice of the people of that State, and bringing to bear upon its Governor and Legislature their demand for immediate righteous action. But it is practically impossible to make the Congressional representatives in Congress of forty-eight States so hot with indignation over matters affecting the citizens of one State only as to unite them for immediate action.[1]

National evils of whatever kind can of course be ultimately cured; but local wrongs, such as I referred to, cannot be promptly corrected by Congressional action. This is simply the verdict of actual experience; and upon it is based the doctrine of "home-rule," which needs no further comment beyond an apology for re-stating and re-defending it before intelligent men. If it were not so old as to be immemorial and so true as to be trite, I might offer abundant proof of it in the wrongs recently suffered by American citizens, without practicability of immediate redress, from the arbitrary acts of Federal officials.

But Mr. Bain need not have prefaced his paper with generalizations about government by emphasis, or criticisms of the inadequate foresight of the fathers, or declarations of the necessity of new methods to meet new conditions. For he proposes primarily nothing which is not wholly consistent with the notions of the said fathers, and with the Constitution which they framed. In other words, he simply suggests that the United States government, owning certain coal-lands in Alaska, and needing for its navy a safe, sufficient, and suitable supply of coal in its own territory, shall open a coal-mine in Alaska. The United States, as actual owner of certain coal-lands, can mine coal from them, if it sees fit to do so, just like any other private owner. Whether the enterprise would be wise from the standpoint either of business or of statemanship, may be a question. But, from the standpoint of constitutional law, it is perfectly legitimate, quite outside of all declamatory emphasis as to social "justice." It is true that Mr. Bain proposes, in connection with this coal-mining operation, further measures, such as the construction of a trunk-railroad, the leasing of other coal-lands to private corporations, and the organization of a selling syndicate, in which the United States government shall be a member. But these other measures might be undertaken, in substance, whether the government had a colliery of its own or not. The first question to be considered, therefore, is the advisability of the mining of coal in Alaska by the United States government for its own use; and this question is to be judged from the standpoint of an owner of Alaska coal-lands who happens to be also a large consumer of coal.

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[1] This is the statement which was reported by the correspondent of a technical journal as a declaration on my part that "you cannot cure national evils," and was characterized as "brutally pessimistic." It is difficult for me to understand how any intelligent hearer could have so misunderstood my declaration; and it is still more difficult to conceive how any man who knew me—whatever he might have thought he heard me say—could seriously ascribe to me a declaration so untrue and unnecessary, and so contradictory to the work of my whole life. In short, I am reasonably sure that I was incorrectly reported, and absolutely sure that I was grossly and unwarrantably misinterpreted.

Where is this colliery to be? The best region for the purpose of naval supply is undoubtedly the Bering coal-field; but every piece of known coal-bearing land in that field is already located, and, as I understand, the rights of the locators have not yet been finally adjudicated. Some of them may be ultimately ousted; others may be confirmed in their titles. Is it certain that the lands of the ousted claimants will include the best location for the proposed government colliery? Should not the government, instead of starting a colliery on lands which it still happens to own, select the most favorable place, without reference to ownership, and buy the tract, if necessary?

The selection of a suitable location involves not only the conditions of mining and transportation, but also the assumed quality of the product. This cannot be settled by the analysis of selected, or even of carefully averaged, samples taken at or near the outcrop. Nothing is more familiar to colliery-managers than the variation in quality—especially in percentage of ash—exhibited by one and the same coal-bed. And every consumer of coal who is also owner of a coal-mine knows that he may lose money by continuing to use his own coal, when it has temporarily deteriorated in quality, instead of selling it for whatever it will bring in the market, and purchasing better fuel for his own special purpose. I was for 20 years connected as engineer with a firm which owned coal-mines, iron-mines, furnaces, rolling-mills, etc., yet often found it economical to sell its own coal and ore, and to buy materials more completely adapted to the other departments of its business.

The plan of a government colliery to supply the navy with coal involves the assumption that the navy will use that coal, even if better fuel can be got. The item of cost is not so important as that of quality; for this affects two vital elements of naval efficiency—the speed of vessels and the distance which they can traverse without re-coaling. It would be the height of folly to make our navy take its fuel from a single colliery. In time of war, the least superiority of one fuel over another might decide the fate of a fleet; and I venture to say that our government would take the best fuel it could find, whatever became of its colliery-investment in Alaska. This supreme requirement of quality at any price distinguishes the navy from ordinary customers.

But, quality being assured, cost becomes important; and a rival fuel, of high caloric efficiency, namely, the oil of California, is now competing with coal on the Pacific coast. The relative cost of the two fuels is worth considering.

Alfred H. Brooks, the distinguished member of the United States Geological Survey, to whom we owe so much of our accurate knowledge of the resources of Alaska, estimates the actual cost of mining in the Bering region at $2 per ton. To one who has had long experience in the Pennsylvania anthracite-region, this figure seems very small. It must represent merely the actual cost in labor and supplies of mining a ton of coal from a colliery completely developed and equipped, without any allowance for interest on the capital invested, or for accidental delays and expenses, or for special exploring-work, or for "strikes"—or, above all, for faults and folds in the coal-bed itself, necessitating new and expensive dead-work, beyond the amount required to prepare a given level for exploitation. Such underground irregularities have ruined many a promising coal-mining operation; and they are characteristic of the Bering field. Mr. Brooks seems to make allowance for them by intimating that the Matanuska coal may prove cheaper to mine, because it "appears to be less folded and faulted." But the Matanuska field is some 200 miles from port; so that the certain disadvantage of the railway-haul counterbalances the possible advantage of the present apparent greater regularity of the beds. In fact, Mr. Brooks puts the cost of Matanuska coal $1 per ton higher than that of Bering coal. But even according to his very low figures for the latter (based on a product of 1,000,000 tons per annum, railroad-freight of only $0.66, ocean-freight of $0.85, unloading-charges of $0.15, and profit to the mine-operator of $0.15 per ton), Mr. Brooks says that Bering bituminous coal should be delivered at the wharf in Seattle at $3.91 to $4.10, and Bering anthracite at $5 per ton.

Now the paper presented at the San Francisco meeting, October, 1911, by Mark L. Requa,[2] declares that "comparing California oil with Alaska coal, it is apparent that oil has complete control of the field." Mr. Requa deems it "safe to say that California oil will dominate the fuel-market on the Pacific coast during the present century, and probably far into the next century." He shows by detailed figures that coal must sell at $3.50 per ton at Douglas Island, Alaska, in order to compete with oil at 60 cents per barrel (the present price being only 30 cents) at the well. Mr. Bain himself admits that "the railroad and the leading coast-industries have now been converted to burn oil imported from California." In other words, a large coal-mine opened at the present time in Alaska, by the government or anybody else, could not be made to pay, unless it were supported by government contracts, appropriations, or privileges—in which case, it would simply be an arbitrary and artificial interference with an established industry of California. From the standpoint of "conservation," the case is still stronger. The superabundant oil-supply of California is irreparably running to waste. Why should this treasure be sacrificed for the sake of the premature development, by artificial support, of a competitive industry?

It may be replied, that the United States Navy must burn coal. But I notice that the latest battle-ship launched by the Japanese is so constructed as to burn either coal or oil at will; and I fancy that we can do whatever the Japanese can do, to meet this problem—especially when our own country is producing and wasting an immense amount of oil, the "conservation" of which would be directly in the line of our most modern "emphasis."

In short, if the government needs fuel for the navy, it had better prepare its ships to burn either coal or oil, and then let the citizens of the country furnish, in free competition, the fuel found most suitable and economical.

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[2] Present Indications in the California Oil-Fields, *Bulletin* No. 64, p. 377, April, 1912.

Mr. Bain's suggestion of an army engineer, or an engineer from the United States Bureau of Mines, as an available manager for this government colliery and railroad, may be dismissed with the simple comment that such officials cannot be spared from the work they are now doing, without putting others in their present places. Consequently, there would be nothing saved by employing them—besides which, it is not likely that Mr. Bain's recommendation would determine the appointment at Washington of the occupant of such a position.

The notion that the proposed colliery could be utilized as a means of testing new methods, or investigating important questions, suggested by the United States Bureau of Mines, is open to two serious objections:

In the first place, no colliery at which experiments of that kind are continually going on, can be kept in operation, as Mr. Bain declares this one should be, in conformity with the conditions of actual practice.

In the second place, no new questions in coal-mining should be tested in Alaska, where the conditions of the industry are, in many respects, certain to prove abnormal. There are hundreds of collieries in the middle West where such novelties can be—and, if plausible, would be—tested, with results far more useful, because more applicable to the average conditions of the industry. One colliery could make place for one promising experiment; but any one colliery doomed to try all the experiments proposed would be nothing but a station, maintained by Congressional appropriations covering its annual deficit, and justified only by its technical, not its commercial, results. Such a station might conceivably be a good thing; but, even in that case, it ought not to be located in Alaska, where results would be less valuable and more costly than anywhere else.

Mr. Bain's further proposals of leases to corporations and a selling-syndicate, including the United States as a member, to control prices, prevent excessive profits and promote social justice, have been carefully elaborated. They are evidently suggested by the policy of the German Empire, which has, in several instances, become a partner in "trusts," in order to be able to control their operations. Concerning these plans, I offer here but two suggestions:

In the first place, the German system, which has its manifest advantages, contains also safeguards against abuses. While I do not personally believe that, as a whole, this "paternal government" is better than our American system of individual enterprise and free contract, I acknowledge the right of others to prefer it. But of this I am sure: that, if the German system is to be imitated at all, it must be copied as a whole, and not piece-meal. We must extend over the whole national service the provisions for preliminary education, secure tenure of office, and pension upon retirement, which we now apply to our army and navy only. They work well, on the whole, in the army and navy; but it does not follow that they could be wisely extended to all Federal functions and offices. This is a question too wide for discussion here. But, until we have deliberately adopted the "bureaucratic" and "paternal" system, it will be folly to adopt single parts of it.

Mr. Bain's paper presents an instance in point. He has devised a careful scheme, which we will assume to be very wise. But every feature of that scheme requires action from Congress, and any amendment of it by Congress may destroy its utility entirely. What security have we, that Congress would not thus fatally "amend" his plan?

Under this head, our best guide will be the record of what Congress has done for Alaska already. The Act of May 28, 1908, "To Encourage the Development of Coal-Deposits in the Territory of Alaska," is a specimen of the legislation to be expected with regard to a distant Territory, and an industry concerning which the members of Congress have necessarily little knowledge. This Act, I believe, was based upon the recommendation of a Commission, appointed by Executive authority; but, in its final form, after sundry amendments, it contained features not proposed by that Commission. Sec. 3, for instance, reads as follows:

> "That if any of the lands or deposits purchased under the provisions of this Act shall be owned, leased, trusteed, possessed, or controlled by any device permanently, temporarily, directly, indirectly, tacitly, or in any manner whatsoever, so that they form part of, or in any way effect any combination, or are in any wise controlled by any combination in the form of an unlawful trust, or form the sub-

ject of any contract or conspiracy in restraint of trade in the mining or selling of coal, or of any holding of such lands, by any individual, partnership, association, corporation, mortgage, stock-ownership or control, in excess of 2,560 acres in the District of Alaska, the title thereto shall be forfeited to the United States by proceedings instituted by the Attorney-General of the United States in the courts for that purpose."

The exact meaning of this verbose provision is not clear; but it has been generally understood to mean that even the holding of stock in any other Alaska coal-mining corporation by a stockholder of a corporation which had purchased coal-lands under this Act, would forfeit the title to those lands. At all events, it is plain that for the least infraction of this vague prohibition, or for any act, not supposed at the time, but determined afterwards by a judge or jury, to be such an infraction, the punishment would be the ruin of the whole enterprise of which the guilty party may have been only an official, agent, or stockholder. We know that stockholders, in consequence of the acts of their directors, managers, or otherwise authorized representatives, may have to pay damages and penalties. But the notion that the doom of immediate, complete ruin should be visited upon a corporation for *bona fide* acts of its agents, subsequently construed to be in violation of a statute, seems to the average stockholder to be confiscation, rather than regulation.

Such drastic measures, however, carry their own cure. They may catch and plunder a few unfortunate investors already involved; but capital has one supreme, conclusive advantage in such a contest: namely, it can run away; and if, fortunately, it has not yet become inextricably involved, it can decline to go in—and there you are!

But the wisdom of Congress did not stop with the measure I have described. The Act of 1908 was really intended, by the Commission which recommended it, to furnish a way in which coal-claims sufficiently large for commercial operations could be legally acquired in Alaska. In provisions not here quoted, it authorizes individual locators under the previous law to consolidate their small claims so as to make tracts not exceeding 2,560 acres in size, and to perfect entry and acquire title to such tracts. This "encouraging" permission, however, was coupled with conditions not previously known to the law. One of these was Sec. 3, already quoted; but, as if that were not enough, Sec. 1 contained an annoying, though probably futile, stipulation as to the stockholders of any corporation thus created; and Sec. 2 gave to the United States government the right to take, for the use of the army and navy, at a price fixed by the President, the coal mined upon the granted tract—the owners being authorized to sue in the United States Court of Claims for an additional price, if they thought they could get it, and were willing to risk the expense of trying.

Of course, such conditions were not attractive to capitalists. At all events, they could not be expected to invest money in Alaska coal-mining, under such risks and restrictions, if there were another way in which such an enterprise could be inaugurated, free from these newly-devised impediments. Such a way was open under the Federal coal-land law, extended by Congress, June 6, 1900, to cover Alaska, which permitted the location of a claim of 160 acres by an individual. A locator under that law, after complying with the prescribed conditions (which need not be recited here), and obtaining his patent, could do what he liked with his own property. He could open a mine on it, or sell it to his neighbor, or contribute it for the formation, together with the similar contributions of his neighbors, of a tract large enough for a long-continued business, such as would invite the investment of capital. By simply waiting until individual patentees were ready to combine in this way, a corporation could acquire their rights for a consolidated enterprise, free from all the burdens and restrictions of the Act of 1908.

The only way (if there was any way) to induce investors to accept the unfavorable conditions imposed by the Act of 1908, was to shut out the alternative offered by the Federal law, extended over Alaska in 1900. It is funny, but true, that this little alternative was overlooked entirely by Congress. The Act of 1908 provided simply an optional procedure, and repealed no previous legislation. Consequently, the former law is still in force; and those who do not like the conditions of the new law can still make valid locations, if they find opportunity, under the old one; so that all former real or supposed evils are continued. In fact, many locators under the former old law are quietly holding on to their rights; and new locations under that law are now and then still recorded. With Mr. Bain's declaration that these locations, whether old or new, should be promptly adjudicated and, if free from fraud, carried to the issue of patents, I heartily agree. But the fact remains, that Congress, with the best intentions, botched the whole business; and the presumption is that Congress would do so again if it got a chance. Congress should not be too severely blamed for legislating in a hurry about matters of which it knew nothing. More blameworthy, indeed, has been its failure for years to remove the incubus imposed by government upon Alaska. But my present point is, that Mr. Bain ought to feel sick at heart, in contemplating the probable fate of his thoughtful plan at the hands of Congress, and the probable shape in which it would emerge from such handling.

Of course, Mr. Bain is not responsible for that result. He has a clear right to suggest any ideal scheme which seems to him desirable. But we others have a similar right to criticise that scheme as impracticable, without even discussing its theoretical merits. I confess, however, that those who make this criticism ought to suggest, if they can, some practicable alternative; and therefore I conclude these desultory remarks with such a suggestion, not based upon adequate personal knowledge, but deserving attention as an alternative to Mr. Bain's costly and otherwise questionable plan.

From my study of sundry reports and maps, I infer that Katalla is the best port for the shipment of Alaskan coal; and I learn that this port and its terminal facilities are now occupied by the Alaska-Pacific Railway & Terminal Co., which has already spent about $500,000 in its enterprise. This company has no connection with the wicked Morgan-Guggenheim syndicate, owns no present or prospective coal-locations, and is simply holding on to its present franchises and constructions until the Bering coal-field shall be allowed, in some way or other, to produce coal. When that desired event shall arrive, this company will be able to transport Bering coal to Katalla by a haul of from 17 to 25 miles, and therefore, at a much lower rate per ton than any other projected line can offer. At present, it is useless to prosecute either this or any other railroad enterprise, while the United States forbids the production of coal. But if coal-mining were permitted in the Bering field, this line, already surveyed and located, and provided with tercilities, could be extended immediately to that field. ıt, the contest between Alaska coal and California oil settled, as all such issues have to be settled, by busi- ıpetition. No doubt there will be a growing local for the Alaska coal; and, in the fight for a market, oal-operators will take greater risks than any manager ernment enterprise would dare to take. Meanwhile, rnment would buy its navy-fuel wherever it saw fit; gress would be relieved of unaccustomed and incom- ole functions.

ld add, that I do not know the exact condition, or the promoters, of the Alaska-Pacific Co., above referred other railway enterprise in Alaska, free from the sus- "conspiracy," would serve my purpose of illustration vell. But this one seems to fill that purpose most ly, having the shortest line to the Bering coal, and ɔeen, moreover, rarely mentioned in the voluminous ngs of the Congressional Committee which harrowed d the affairs of Alaska. Such scanty notice from such l is almost a certificate of merit!

AM H. HAMPTON, Katalla, Alaska (communication to etary *):—On the basis of long observation and inti- sonal knowledge of Alaskan conditions, I desire to say great necessity of Alaska is facilities for transporta- ch, to a considerable extent, must precede the develop- the region in other respects. Capitalists are not eager ınd equip large mines, unless they can be sure of the which will bring their machinery and supplies, and ay their products. On the other hand, pioneer rail- are not attractive investments, unless they can be ly assured of a profitable business from the new terri- ch they penetrate and develop. There is thus a dead- ich is, however, more apparent than real, if only let or, as experience has shown everywhere on the Pacific modern American system of running railways into ped regions (as distinguished from the old system of ng by rail the centers of trade already established) is e set in operation, if the opportunity be given to it. Alaska, the United States government has made this

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* Received June 12, 1912.

apparent dead-lock real, by placing artificial restrictions upon both railways and mining. And, the development of Alaska having been thus made at present impossible, various experimental remedies, such as government leases, or direct government operations, are suggested to relieve the situation which the government itself has created. In other words, new experiments are proposed to cure the evils caused by old ones.

Now, I do not think it is wise to try such new experiments in a new country like Alaska, which needs all possible fostering encouragement in the exploration and exploitation of its resources, and cannot bear any additional risk or burden. Nor can I see that such extraordinary measures would be necessary, if certain obviously just and proper things were done.

1. Let the laws be obeyed by the officers of the government, as well as by citizens; and let the citizen who has obeyed the laws be free from arbitrary interference, and, above all, from the stigma of fraud, placed wholesale upon an entire population, as a pretext for the denial of rights conferred by statute. Congress has invited the citizens of the United States to engage in the difficult enterprise of settling and developing Alaska upon certain conditions. Those conditions have been fulfilled by many citizens in good faith; and the United States has received $360,000 in payment for coal-lands, located under the law in the Bering field. Yet not a single United States patent has been issued in that field. Some of the claims are disputed by the government, it is true, and on the merits of that dispute I say nothing. But with regard to all of them, disputed or undisputed, patents are arbitrarily withheld; and claimants are informed that the money they have paid in good faith cannot be returned to them except by special Act of Congress. When, if ever, that special Act shall have been passed, they may get their money back, without interest—for Uncle Sam pays no interest on "claims." Apart from all questions of law, is this the way to encourage the settlement and improvement of a new—and especially an Arctic—territory?

2. But these claimants may have some legal rights—a matter yet to be finally determined. Meanwhile, no new titles can be granted in the Bering field until these have been finally either confirmed or extinguished. Hence, the obvious duty and policy of the government is to settle these titles as soon as possible; and, if this duty were promptly discharged, there

no need of other and dubious means to break the k.

gislation for the prevention or regulation of "trusts" onopolies" is all right. Only, let it be legislation, and rary Executive interference. Let the law itself fix the 1s to which individuals and corporations are to be and let them not be left dependent upon Executive nd Department regulations, fixing new conditions. I at it is already the privilege of a citizen to resist illegal legal regulations, and to insist upon his rights under But the privilege of fighting the government is too e for a poor man—and rich men are notoriously unto buy, together with a right, the risk and odium of Moreover, the poor man is forbidden even to sell his of battle before he has won a victory!

the Matanuska coal-field, very little money has been paid to the government for coal-lands. Most of the ms, I believe, have been canceled. This explains the attention lately called to the subject of government or the development of that field, and in connection h, the plan of "unloading" upon the government, as line, the old Alaska Central railway. A bill has been introduced in Congress, providing for the assumption nterprise by the United States, and the transfer, from mus of Panama to Alaska, of the machinery, material, required to complete the railroad! If it be really necessary to undertake such an enterprise, the governght certainly to be free to choose the place for it, and orced to select this field by having excluded, through unnecessary and still revocable action, the Bering field. sides the known high-grade coal of the Bering and Mafields, all good, bad, or indifferent coal-lands, with an 12,000 sq. miles, have been arbitrarily withdrawn from ion and location. Within this area whoever mines a coal will be pursued as a criminal by United States agents." A still larger tract, established as a "forest-" is an additional bar to the industrial progress of and the enterprise of its citizens. No wonder the situas created, and prolonged for years through Executive nd Congressional inaction, calls for relief. But the ion of relief through further interference does not comself as wise.

6. Evidently, Mr. Bain has not been on the ground in person; I therefore beg to correct an error into which he has fallen, concerning distances, routes, and connections. Namely, the line of the Copper River & Northwestern railway, for the transportation of coal from the Bering field to its terminus at Cordova, passes at Katalla the terminus of the Alaska-Pacific Co.; and its extra length, beyond Katalla, is not 58 miles, as stated by Mr. Bain, but 75 miles in an air-line. The latter road strikes the first Bering coal 17 miles from its terminus and harbor; and at 25 miles it is in the heart of the Bering field. From a point on this direct, short route to the coal, a cross-line of about 26 miles would connect with the Copper River & Northwestern for purposes of interior trade. Having personally made, before the advent of the C. R. & NW. Co., the survey of much of the line now occupied by that company, I make this statement upon positive knowledge.

7. In conclusion, I beg to repeat that, in my judgment, what Alaska needs, and has a right to expect, is simply fair play, such as heretofore has been given to other Territories. Mr. Bain suggests that his plan of a government mine, a government railway, government leases of coal-lands, and a selling-syndicate, of which the United States shall be a member, be applied to Alaska only at present, and afterwards, if the results prove satisfactory, extended to the remaining coal-lands in the States. As a citizen of Alaska, I would urge, on the contrary, that this complicated experiment be tried first on some patient not already moribund. If the results should be satisfactory, and if Alaska should be permitted meanwhile to recover strength without the interference of doctors, she might be able to stand the new treatment, when it had been elsewhere demonstrated as successful.

As to the general policy of leasing public mineral lands, I would only say that, if it should be adopted, I think the Canadian system might well be followed. Under that system, the government gives something, in the way of the efficient administration of justice, the protection of property, public improvements, etc., for the royalty it exacts. The mere acquirement of rent or royalty, without such compensations, is nothing but the old Spanish colonial system, which promotes tyranny and ends in ruin.

TO REVISION.

NSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS.]

The San Nicolás Mining-District, San Nicolás, Tamaulipas, Mexico.

VING H. WENTWORTH, MATEHUALA, SAN LUIS POTOSI, MEXICO.

(New York Meeting, February, 1912.)

little town of San Nicolás, Tamaulipas, Mexico, lies imately 60 miles east of Linares, situated in the State of Leon, and 150 miles SE. of Monterey, as shown in the -map, Fig. 1. The place is reached from Monterey by the co train as far as Linares, a distance of 92 miles, and y coach the rest of the way. It is situated in a narrow between low hills, the houses forming one main street plaza. It is peopled by about 500 inhabitants.

climate is temperate, the thermometer ranging from 90° mer to a few degrees below freezing-point in winter; s severe weather only lasts a day or two at a time, when orthers" blow.

e are no streams in the vicinity, but water sufficient for of the people is obtained from wells. The rainfall is the same as that in the rest of NE. Mexico. The rainy s are July, August, and September. The rest of the practically dry, with an occasional showery day or two the winter, when a "norther" is blowing.

ber is abundant in many localities. There are two varie- oak, besides mesquite, ebony, "maguira," "patol," and ber of other kinds of trees indigenous to this region.

I. History.

he discovery of the silver-lead bearing veins occurring district, the mining-camp of San Nicolás was founded 7 by the Spaniards, and numerous mines were opened orked for a distance of about 3 miles east and 5 miles the town. At the western extremity and near the Car- mine, a small town called San Pedro was built. The walls of the houses form a single street, and all is over-

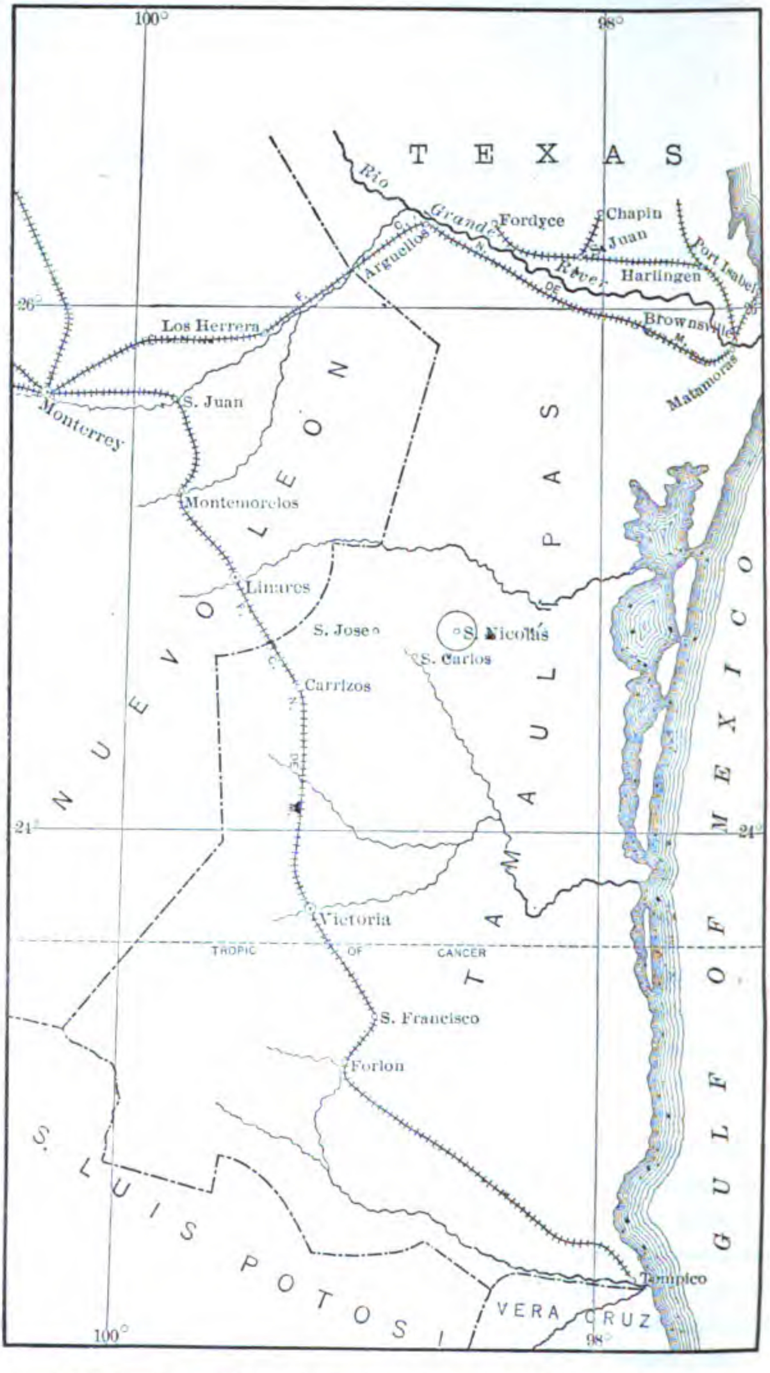

FIG. 1.—SKETCH-MAP OF THE STATE OF TAMAULIPAS, MEXICO.

ith brush and small trees. A little farther east of this the Santa Bárbara mine are the ruins of another set- Judging from the old records preserved in the of San Nicolás, together with the innumerable mine- scattered along a line 8 miles long, great activity was l in those early days, and the population grew to habitants. All was suddenly brought to a stop in 1810, r broke out with Spain, resulting in the independ- the country. The political disturbances which fol- rapid succession up to about 35 years ago, together fact that the country was over-run by wandering hostile Indians from the north, prevented further de- nt of the camp.

ystem of mining employed by the old Spaniards here same as that carried out in many other parts of the viz.: small irregular openings reaching to no great The ores were treated by smelting in small furnaces, es at the mines, and sometimes at some distance away. tes are marked by piles of slag, some of which are to as far off as the town of San Carlos, 18 miles SW. from olás.

all appearances, the location of these *galemes*, as they d, depended on the supply of water, or wood, or both ; for a particular kind of wood was used, called *ocotillo.* nall tree that grows throughout this part of the country. od is hard and burns with a fierce heat.

e opening of the Monterey & Mexican Gulf railroad art of the Mexican National system) through to Linares lose of 1889, much needed means of communication ovided with the Monterey smelters. Mining-operations newed by progressive individuals in some of the old es, and new ones were opened. Further facilities were to the district by the building of the Linares & lroad by the San Carlos Copper Co. from Linares to é in 1906, which shortened the haulage in carts from miles. The traffic on this road, however, was stopped ary, 1911, by the closing down of the company and of its premises.

II. GEOLOGY.

In traveling from Linares to San José, one crosses first a broad rolling country well timbered with mesquite, and beneath the alluvial soil lie slates and shales of Jurassic age. After covering two-thirds of the distance and on approaching the foothills of the San Carlos mountains, one comes to Tertiary conglomerates, the débris of the Cretaceous formation, which soon follows. This latter covers the entire district that is the subject of the present article, and may be divided into three distinct parts, each with marked characteristics. The lower part is composed of approximately 200 m. of thick-bedded gray limestone containing 94 per cent. of $CaCO_3$, the middle part of 50 m. of a thin-bedded siliceous limestone from 4 to 12 in. thick, streaked with bands of chert, and the upper part of slates, impure limestone, and shales, about 200 m. thick. One class of rock changes abruptly to another without any slow gradation.

The San Carlos mountains form a rugged ridge extending south from San José for a distance of 12 or 15 miles, and rise to a height of 4,000 ft., more or less, above the plain, or about 5,000 feet above sea-level. Fifty miles east, there is a similar ridge running parallel to the first; and in between the two, and extending to the north, the limestone, with a breadth of 20 miles, rises to an average elevation of 2,500 ft. above the sea-level.

This is deeply furrowed by canyons over the whole extent, with drainage N–S., in which directions it slopes gradually from the center. Crossing this formation are a few wide folds with gently-sloping sides, whose axes run N–S. Besides these, there are other folds running E–W., which are short and sharp, forming monoclines and synclines. It is along the latter that the limestone has been fractured, and mineral-bearing veins have formed.

The country has undergone two different periods of igneous intrusions during Tertiary times. In the first, large bodies of syenite penetrated the limestone at San José and to the south, altering the surrounding rock in places to marble, and also forming garnet and other minerals peculiar to contact-metamorphism. It is along these contacts, and especially in close relation to the garnet, that the copper-ores in the mines at San

ire found.[1] In the second period, a large number of
and laccoliths of basalt were formed throughout this part
country. The limestone, which has just been described,
between the two ridges of mountains, was elevated dur-
is epoch, forming a broad dome. Below this dome, there
btedly exists a large laccolith of basalt, for the center is
d with a large mass which forms the Sacramento moun-
single peak with precipitous sides, rising 500 ft. above
mestone. This peak is situated 3 miles west of San
s.

nerous dikes and small laccoliths are scattered over the
limestone-area, having their source probably in the main
th. These intrusions have altered the limestone but lit-
t indirectly they have formed the means of the minerali-
of the district.

have thus a limestone-district folded and faulted by
is intrusions; and the fractures resulting from these
nents became the water-courses through which the warm
ons in their upward flow have formed metalliferous veins.
principal mineral is argentiferous galena, with a little
carbonate and sulphate occurring generally as scales
d the sulphide where the latter has been oxidized. Occa-
ly, small amounts of blende occur associated with the
. The gangue-matter of the veins is barite, calcite, and
ne mines quartz. Often the ore is imbedded in a hard-
clay derived from the adjacent rock by faulting. In the
part of the veins, silver chlorides often occur in minute
les disseminated through the gangue; and while in most
the amount is insufficient to be of economic interest, ow-
present conditions, still paying values are often encoun-

bulk of the ore recently mined in this district has been
from the siliceous limestone, which forms the center of
ormation. The mineralization has been effected by re-
nent of the limestone by the minerals brought up from
by warm solutions. These waters have been arrested in

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[1] F. Kemp, The Copper-Deposits at San José, Tamaulipas, Mexico, *Trans.*,
178 to 203 (1905). G. I. Finlay, Geology of the San José District, Tamau-
lex., *Annals of the New York Academy of Sciences*, vol. xiv., pp. 247 to 295
3).

their upward course in a number of localities by impervious or semi-impervious strata, and the solutions have penetrated along the bedding-planes of the limestone just under the slate, forming blanket-veins, some of which are 15 m. wide. This restriction in the upward circulation of the water has caused the ore to have a great tendency to follow certain strata; so that instead of the vertical or oblique ore-shoots, such as are predominant in nearly every mine, there are horizontal ore-bodies. One characteristic of these ore-bodies, and a favorable one at that, is their regularity and extension. On the Todos Santos vein, these shoots have been followed for 1,000 m. with the ore still continuing; and in the Moctezuma mine, the vein has been explored 500 m. without finding any change. The veins penetrating the slate that composes the upper part of the formation are narrow, and the ore occurring in this rock is pockety. The narrowness is due to the fact that the slate possesses more resistance than the limestone; so that the fractures narrow down immediately after passing above the siliceous limestone, become ill-defined, and then, in most places, disappear before the surface is reached. In the immediate neighborhood of San Nicolás and in this slate, the galena occurs in quartz, which in many places forms bands along the walls of the veins, while the center is filled with calcite which is practically barren. Farther west, however, at El Palmar and Moctezuma, the veins in the slates consist almost entirely of calcite, carrying galena and silver chlorides. In the extreme western part of the district no quartz is found.

III. VEINS.

The Todos Santos Vein.—This vein passes through the Mariposa, Todos Santos, Cuatro Vetas, La Condesa, and El Conde claims, which are located in the extreme western part of the district, as may be seen in the accompanying map. Its course is 70° NW–SE. astronomical, and follows a fold in the rock that in some places forms a syncline, while in others a monocline, see Fig. 2. Most of the ore occurs on the south side of the trough, in some places dipping north, but the general dip is to the south at an angle of from 70° to 80°. The ore in the vein varies from a few inches to 5 or 6 ft. in width, averaging about 2 ft. The blanket-veins, or *mantos*, as they are called composed of almost pure galena, varying from 1 to 8 wide, but 5 or 6 in. is about the average width. At tion of the vertical vein and the *mantos*, the ore in es widens out considerably. The blanket-veins are ell defined, but the vertical vein has no regular walls, enetrating the limestone at variable distances, with a to form small *mantos* along the bedding-planes.

orkings on this vein have uncovered two *mantos*, the one being just under the slate, and the other one m. below the first.

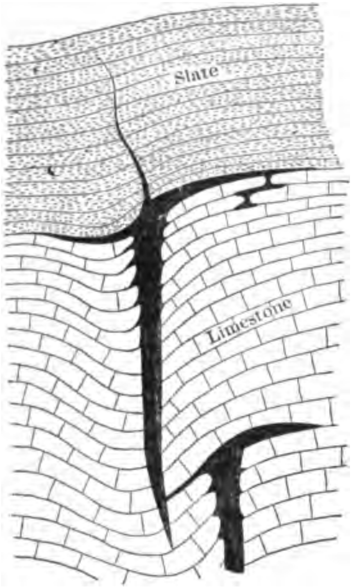


FIG. 2.—TODOS SANTOS VEIN. CROSS-SECTION.

ticline passes through the center of the Todos Santos th its axis running N–S., the formation dipping westough the Mariposa and beyond for a distance of 8 or and eastward about 6 km. to El Palmar. An *arroyo* down the top of this anticline, exposing the thickgray limestone. The dip of the formation both east t is gradual; and although in places the angle is as 15°, the total dip to the east at the end of the 6 km. is 0 or 125 meters.

ein penetrates the slate above the siliceous limestone ort distance, and consequently only shows on the surface near the Todos Santos anticline. On account of the difficulty of locating the vein, nothing is known of it beyond the workings on the claims above mentioned.

The Moctezuma Vein.—This vein lies about 500 m. south of the Todos Santos vein, and is parallel to it. It passes through three claims, the Andrea, the Moctezuma, and the San Andrés; and the camp is known as Moctezuma.

It is a stronger vein than the Todos Santos, as it penetrates the slate to a greater distance above the siliceous limestone; but the *mantos* that have formed on it are small and irregular. Still it is to be noted that just under the slate the ore is wider and is of a better grade for some meters in depth than farther down, thus forming a horizontal shoot.

The dip of the formation here is to the east, the same as along the Todos Santos vein. Owing to the difficulties in prospecting here, nothing is known of this vein either east or west of the camp.

The Carroleña Vein.—The next known vein is the Carroleña, lying between 400 and 500 m. south of the Moctezuma, and running about 73° NW–SE. It passes through the Santa Claus, Maria, and the Carroleña claims, and then turns S. 53° E. into the Dulcinea; thence east into the Don Quijote and San Nicolás, where it makes another turn to S. 35° E., and passes through the Santa Bárbara and Desamparados.

The Desconocida Vein.—This vein is parallel to the Carroleña, and lies about 75 m. to the south. It traverses the Desconocida, Dulcinea, Don Quijote, and the rest of the claims crossed by the first vein.

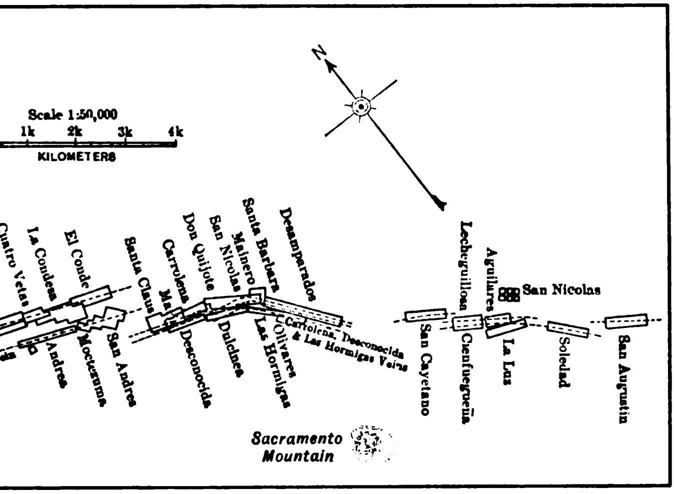
The Hormigas Vein.—About 100 m. south of the Desconocida is the Hormigas vein, passing through the Dulcinea, Hormigas, and Olivares claims.

Taking the three veins together, they are similar to the Moctezuma, being strong fractures, penetrating far up into the slate, and can be seen on the surface in many places. To the east on the Santa Bárbara and Desamparados claims, they combine the features of the Todos Santos vein by forming *mantos* just under the slate. Their distorted course is due to local horizontal faulting. The mines located on this group of claims form the camp known as El Palmar.

s been mentioned before, the formation dips east from antos to El Palmar; thence it rises towards the east to just beyond the Desamparados, where the lower lime-ows again on the surface.

urther, the country dips again slightly to the east on town of San Nicolás.

d the Desamparados are a number of claims strung out st, as shown in Fig. 3. These claims are located along tinct parallel veins, which outcrop through the slate. efly along these outcrops that the old Spaniards worked. a probability that these three veins are the continua-the El Palmar group.



-MAP OF SAN NICOLÁS MINING-DISTRICT, TAMAULIPAS, MEXICO.

IV. ORE-TREATMENT.

the renewal of mining-operations in this district, the e been treated by hand-concentration. First, the clean lena is picked out of the ore as it comes from the d the rest is then passed over a 1-in. or a 0.75-in. reen. The coarse dirt is ground in Chilean mills, gyra-jaw crushers, and, together with the fines, is washed *llas* to get rid of the clay. At the same time, the bulk ed more or less to two-thirds by throwing away the light rthless dirt that is carried down to the bottom of the *planillas.* After that, the ore is jigged by hand in a box from 3.5 to 4 ft. long by 15 or 18 in. wide, with a 4-mesh screen in the bottom, submerged in one end of a box about 5 ft. square, which contains the water. The jerking motion is produced by means of a pole from 12 to 15 ft. long.

The large box is divided in the middle by a baffle-board that reaches nearly to the surface of the water. Most of the ore fine enough to pass through the 4-mesh screen collects in the compartment under the jig-box, while the motion of the jig throws the light and worthless slimes over the baffle-board into the next compartment, where they collect.

The ore thus treated gives a product that assays as follows: Silver, from 1,500 to 3,500 g.; lead, from 60 to 70; insoluble, from 8 to 12; iron, 0.5; lime, from 2 to 3; and sulphur, from 5 to 8 per cent.

The camps that have been producing during the last 20 years are the Todos Santos, Moctezuma, and El Palmar. The other mines farther east have been operated but little. On all the properties, the workings are comparatively shallow, the shafts ranging from 40 to 125 m. deep.

The lower limestone has never been explored by any of the companies, owing principally to the depth necessary to undertake the work.

The miners have contented themselves in working along under the slate at small depths, although in this lower limestone there is a probability of encountering large deposits of ore along the fractures.

REVISION.

CTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS.]

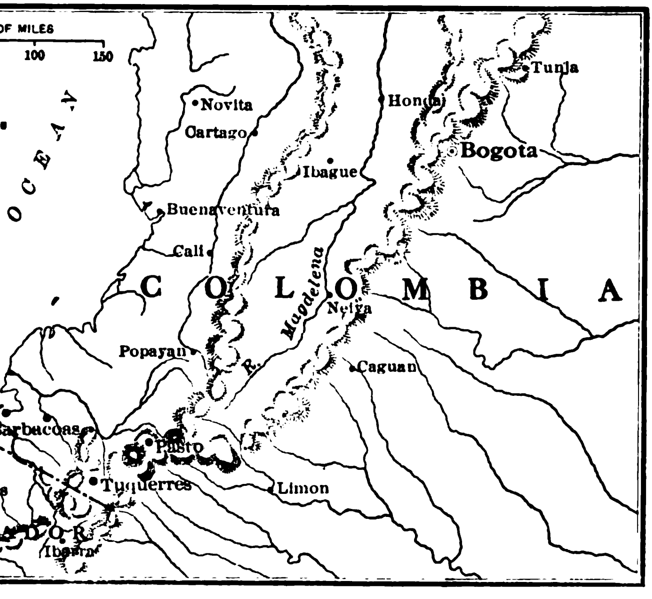
Gold-Mines in Southern Colombia.

RTUNATO PEREIRA GAMBA, TUQUERRES, COLOMBIA, S. A.

(San Francisco Meeting, October, 1911.)

I. GENERAL DESCRIPTION.

epartment of Nariño (formerly included in the De- of Cauca) is a narrow band about 170 km. (100 le in the southern part of the Republic of Colombia,



—SKETCH-MAP OF THE SOUTHWESTERN PART OF COLOMBIA.

boundary of Ecuador, as shown in the sketch-map, ts capital is Pasto, a small city of 25,000 inhabitants; pal towns are Túquerres, Cumbal, Ypiales, Barbacoas, aco. The port on the Pacific ocean is Tumaco, and he interior is carried partly on the Patia river (steam to the fluvial port of Barbacoas), and afterwards over poor roads by human bearers or on mule-back.

The weight of packages is usually limited to 100 kg., and the freight-cost for heavy pieces is enormous.

The climate is tropical: warm in the lowlands and cold in the highlands; the variation in temperature being about 1° C. for every 160 m. difference of level. The plateau and highlands are very healthy.

Unskilled labor at the mines is principally furnished by Indians, and is first rate; moreover, these Indians and the *mestizos*, once taught, are excellent miners.

The wages are from $0.50 to $1.50 and board (costing $0.50) per man per day, reckoned in silver at about $2.50 per $1 American currency. But both wages and the cost of food vary with the locality, climate, distance from sources of supply, etc.

Vein-mining is of very recent date here. There is no tradition of its existence in Spanish times. On the contrary, the alluvial deposits on the coast (Barbacoas, Mayasquer, Sanabria, etc.) were vigorously exploited, and yielded much gold. These alluvial deposits are very productive just now, being worked unsystematically by scattered parties of negroes.

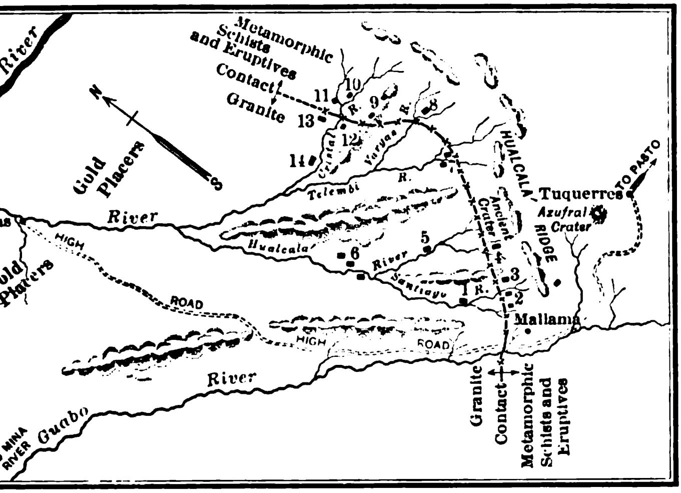
The predominance of eruptives is the most remarkable feature of the geology of the country; the sedimentaries are, in most places, yet undiscovered. The most common among the eruptives are andesites and basalt-like andesites; these greatly interfering with the metalliferous deposits. Besides the volcanic rocks, igneous rocks of the granite type are very common.

All mining-operations here have gold as an object. Copper-deposits have been found; but no attention is paid to them. Mining in Southern Colombia is still in the prospecting stage. Not a single mine is sufficiently worked or developed to furnish thorough information; but the industry is growing rapidly in importance.

II. The Gold-Belt.

The gold-bearing zone has been explored for a length of 50 km. (30 miles) in a general N–S. direction between the districts of Mallama and Samaniago, the southern part being Mallama. It lies on the west side or Pacific slope of the western Cordillera of the Colombian Andes. Fig. 2 is a sketch-map of the Mallama and Samaniago districts.

It is my belief that most of the mines of this belt have had a ı origin, and there is complete evidence that they have rmed by descending waters. In some of the mines the of formation can be observed. The differences which ıalize the mines must be ascribed to the country-rock the presence in it of certain minerals (*e.g.*, manganese- 'he hypothesis of expiring volcanism (of Spurr, if I am taken) receives complete confirmation here.



MINING-CLAIMS.

ıllama District.
1. Alaska.
2. Porvenir Group.
3. Alegria.
4. S. Pablo.
5. Bombona Group.
6. Gualcalá Group.
7. Villas Group.

Samaniago District.
8. Redencion.
9. Rocio Group.
10. Concordia.
11. Socorro.
12. Madrono.
13. Selva.
14. Canillera Group.

.—Sketch-Map of the Mallama and Samaniago Districts, Colombia.

extinct volcanoes, whose craters are still traceable on und, are one of the most striking features of the belt. probability these volcanoes were active at a period not ı the present. The eruptive waters, carrying the min- s, ran upon the surface of the ground, soaking and im- ting the country-rock, and at the same time filling the s and joint-planes of the formation. Such is, in syn- the most plausible hypothesis to explain the special character of the majority of the deposits of the belt from Mallama to Samaniago, and other very similar deposits in this country.

In general, the mines of the belt present two types of ore-deposits: 1, the *mantos*, that is, the impregnated rocks transformed into an earthy (saprolitic) material; and 2, the veins, more or less regular in formation.

According to their mineralization, these deposits can be divided into two groups corresponding also, in a certain measure, with their geographical position: (A) The Mallama (southern) group of pyritic ores; and (B) the Samaniago (northern) group of free-milling ores.

In all the veins the gangue is quartz; among the sulphides mispickel predominates, and afterwards common pyrite; then blende, chalcopyrite, and galena in very small quantities, and the telluride of gold rarely. (Some deposits in the granitic rocks characterized by the presence of molybdenum-ores belong certainly to another order of formation, but there are not sufficient data for a detailed description. A superposition of the two is observed in certain localities).

The southern end of the zone is occupied by granites, syenites, and monzonites, one rock passing into the other, but with predominance of the syenitic type. I will call the whole series granites. All these rocks are highly hornblendic, very frequently porphyritic in structure and rich in zircon.

This granitic formation has been covered in great areas by greenstones (augite-andesite, which I have called prophylite in my Spanish writings about this metalliferous zone), ejected from many of the volcanoes now extinct. These basalt-like andesites form many varieties within a common type; and to them I ascribe the origin of the ore-deposits.

In the northern part of the zone clay-schists (lying, as we may conjecture, on the granites) are the country-rock on the top of the hills, passing into slate at the base. Dikes of light-colored andesites, entirely kaolinized, can also be observed.

Complete geological information cannot yet be obtained, since the country is quite unexplored, and covered with woods; but what I have already said may serve to give an idea about the most important points.

pographically, this belt is the most distorted, broken, illy country in the world.

III. The Mines.

re than 2,500 claims have been located in this zone; but few of the mines are equipped or even explored. The min- w in Colombia does not compel the miner to perform lly any given amount of work on his claim. To hold the rty, it is enough to pay (very moderate) taxes; and, there- a great deal of the mineral land is owned by people who t work the mines, and the underground richness is prac- y as non-existent.

The Mallama Region.

this region two groups of mines are worth describing:

The Porvenir Mines.—These deposits occur at the contact een granites and greenstones, Fig. 3, occupying a very iar formation of veinlets in the granite country-rock, and ng a workable mass of ore about 30 m. (100 ft.) wide. verage tenor of gold is $9 American to the ton. All ore- s will be given in United States currency. There are also w veins near the contact running into the eruptive; these gers are highly pyritic and rich. The sulphides are prin- y mispickel and pyrite intermixed, and associated with quantities of blende, chalcopyrite, galena, and tellurides. e pyritic ore frequently assays $238, and the average of e vein-stuff $47 to the ton. These values are principally ld (about 30 per cent. being silver).

e Porvenir claims are the highest in the belt, the upper- workings being 3,400 m. (11,300 ft.) above tide-level. limate is very cold, and healthy. As the claims cover a territory and the slope of the Cordillera is very steep, there ithin a short distance lowlands with very agreeable cli- and well provided with timber and water-power.

e property has been explored a little under my direction, small plant (three-stamp mill and concentrator, made by . Krupp) is now under erection. The intention is to de on a very small scale, after roasting the rich pyrites oncentrates.

laboratory experience in cyaniding these ores was as follows: After crushing to 60-mesh, I extracted from crude unroasted pulp 80 per cent. of the gold contained in material assaying $9 per ton, and 50 per cent. from $240 stuff. From the latter, after roasting, I extracted 95 per cent.

The free gold in these mines is limited to the outcrops and saprolitic ores; but I think experience with the stamp-mill will prove that there is a great deal of it in the pyrites of secondary enrichment, though no important amount of "colors" can be obtained from them by panning.

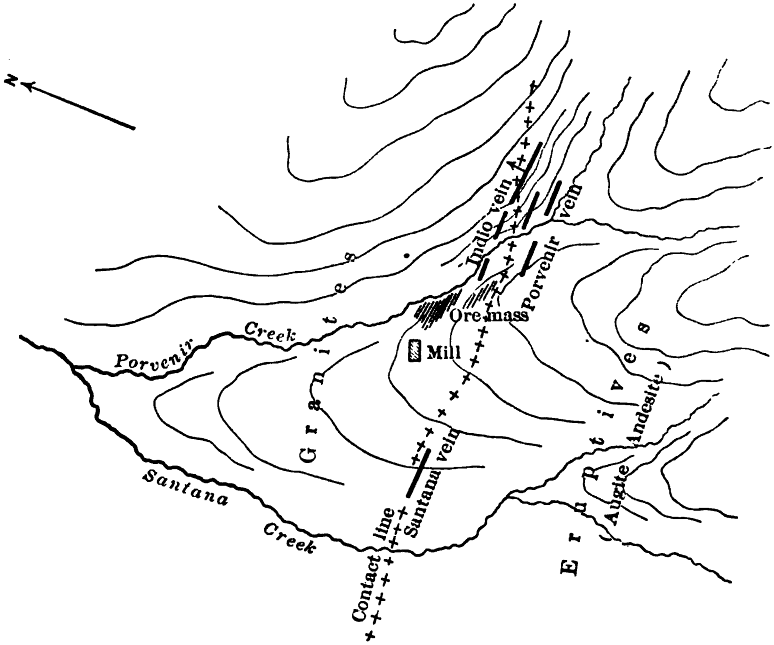
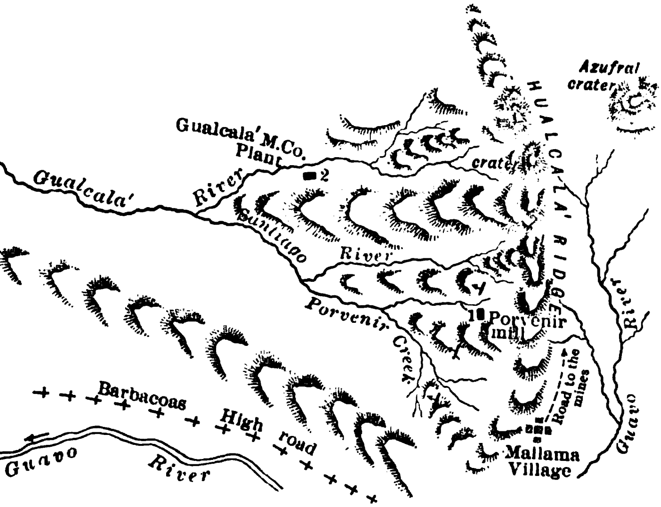


FIG. 3.—THE PORVENIR MINE.

The general direction of the system of parallel veins and veinlets is E–W., and the dip N. from 50° to 60°. The thickness of the veins is very variable, they being lenticular masses, but an average of 0.50 m. (1.5 ft.) can be established, maybe something more. Great irregularities are to be noted, both in formation and in mineralization.

This property is one of the most interesting in the region, and when more fully explored will throw great light on the study of the genesis of the ores. Its formation depends immediately on the extinct volcano of Hualcalá, the crater of which the boundaries of the property. (Following the Inca phy this name is written with H, but the official name alá.)

Gualcalá Mines.—Very near the Porvenir group, Fig. t a lower altitude (2,700 m.), an American company ery important group of mines, and has erected an up- ll-sliming cyanide-plant. Col. R. A. Linton, an Amer- ineer, is in charge of the enterprise.



Sketch-Map Showing the Position of the Porvenir and the Gualcalá Mines.

roperty is a system of parallel lodes, some of them very ned and regular in formation. The Bombona vein has st explored, and is somewhat extensively developed in els. It is a clearly-defined fissure-vein of an average s of 1 m., with a gangue of white quartz and shoots of metal"—a complex of mispickel, galena, blende, and very rich both in gold and silver. I imagine that the value of the whole of the vein-stuff may be $30 a ton; , very high assays are obtainable.

vall-rock is monzonite (which has been called also iorite); at the top of the hill greenstones are found, possible that the same formation as at Porvenir exists The Gualcalá Mines Co. has the merit of being the first luce the most modern methods of metallurgical treat- to southern Colombia.

The mineral deposits of this part of the belt are subordinated to the extinct volcano of Hualcalá (Gualcalá). This crater, doubtless, was the most important in this part of the country; its ejections covered immense areas, and the crater itself, as it is traceable on the ground, is of enormous dimensions. Some of the remains of its walls form the very picturesque obelisk-like *Pico do Gualcalá.*

The volcanic breccia covers almost all the surface of the territory. This material acted as a filter for the mineralizing waters. It is observable that this breccia is something like "the mother" (*madre*) of the ore-deposits. In the Porvenir mines, for instance, it is easy to see the veinlets formed by the percolation of the materials of the vein through this breccia or *madre.*

The Samaniago Region.

This region attracts attention especially by its free-milling ores, which are historically and economically worthy of interest, as comprising the first worked and the now productive deposits of this part of the Republic.

I shall limit my description to a few of the ore-deposits now operated, especially the Concordia mine, which was the pioneer enterprise. Being now consulting engineer of this property and all its neighbor mines, I am able to furnish trustworthy information concerning them.

1. *The Concordia Mine.*—The country-rock is a clay-schist, altered, on the top of the hills, passing into "writing-slate" at the base of the formation.

The valuable mineral deposits are confined to the upper part; the more saprolitized the schists the better. These clay-schists, in the "good places," are something like natural brick.

The two kinds of deposits already mentioned, *mantos* and veins, are very distinguishable here. The values in the first are somewhat erratic; but the whole mass contains gold from $0.66 per ton to very high grades.

The veins are very peculiar: not sharply defined; very irregular in thickness; sinuous in trend, and generally low in sulphides. The principal lode is a flat vein, reinforced by numberless feeders going from the *mantos* to the vein.

In the rich spots the quartz is smashed as if it had passed through a crusher; and, in general, the gold is loose, not ad-

:o the gangue—a feature peculiar to these ores, in which 'y rare to find specimens of quartz showing gold, even extracted from places as rich in free gold as from $600 per ton.

batea is in this locality the most useful assay-outfit for ning rapidly the quality of the ore, specially for the material of the *mantos*. With some practice, it is very y the aid of this apparatus, to ascertain the values

ably the veins here belong to a compound system by a principal, much distorted, flat vein with feeders ystem of secondary parallel veins. Many of these had rmed before the time of the ore-forming eruption, and of them were enriched by that agency. This explains art of the irregularities observed in the distribution of ıes, and the existence of numberless quartz lodes prac-barren.

rule, the paying veins are narrow, from 0.30 m. to 0.60 :. to 2 ft.) thick; commonly ribboned, but most gener-ıashed; charged with manganese oxides, and with a n sulphides of from 1 to 2 per cent. The big lumps of ıartz are of very low grade; the same remark must be to the big lodes.

ng the sulphides mispickel predominates, together with lack metal." The tellurides, though not yet isolated, een recognized by their reactions and are common in her ore. The "headings" from concentration show riable assays, from $600 to $2,400 per ton. The bullion es 780 in fineness.

mine has been badly "picked" in past years by min-y the rich vein-outcrops and the richer part of the *man-*ardless of future development. The average value of :es in the past was from $30 to $50 per ton in free gold. e situation is changed, and the mine must be worked and treated by scientific process; but the property still holds e as a very rich and profitable one.

metallurgical plant has been the crudest that could be ed: wooden native mills and silvered copper plates, with-y attempt at concentration or provision for further y save a very perfect storage of the tailings. At present a 5-ft. Huntington and a 10-stamp mill, made by Fraser & Chalmers, England, are under erection. The property will probably pass into the control of the American "Gualcalá Mines Co."

The location of the Concordia, Transval and Socorro mines is illustrated in Fig. 5.

2. *The Socorro Mine.*—In general features, this resembles the Concordia; but the deposit has been worked more systematically, principally by open quarrying, with little underground mining.

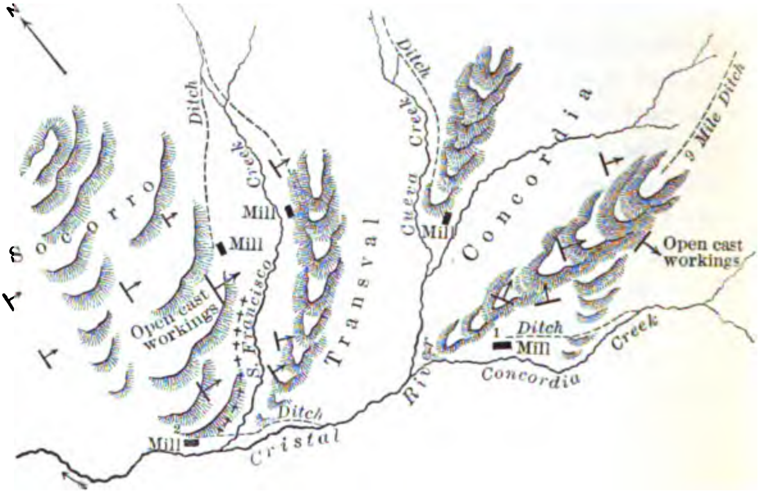


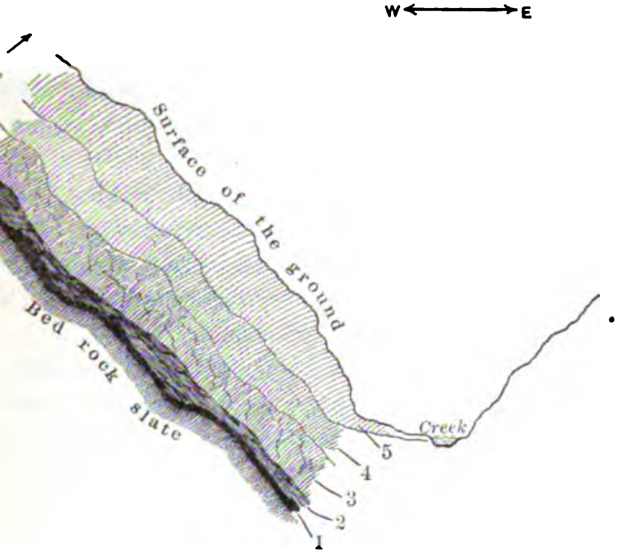
Fig. 5.—Sketch-Map Showing the Position of the Concordia, the Transval, and the Socorro Mines.

The *manto* formation predominates here. The ores are very clayey; manganese is not so common; and no remarkable amount of tellurides is present; and besides these differences from the Concordia mine, there is a greater parallelism and regularity of the general trend of the veins, which is very constantly E–W.

The proportion of concentrates from the milled ore is 0.3 per cent., of an almost constant value of 1 kg. of gold ($667) per ton. This low tenor in concentrates depends on the fact that the greater part of the milled ore is saprolite and shaly material from the *mantos*.

The quartz veins are narrow (maximum thickness, 0.40 m. =

eautifully ribboned, and not so crushed as in Concor-
ey carry about 4 per cent. in sulphides.
etallurgical plant is a native mill.
en-cast workings of this mine are highly interesting
tudy of the geology of the ore-deposit, Fig. 6. The
out 80 m. high from the level of the creek, and follows
of the vein. The schist of the wall-rock is hard and
the dip from 60° to 70° E.



1. Quartz.
2. Ocherous material. Very rich. } *Manto.*
3. Saprolite.
4. Clay-Schist.
5. Soil.

6.—Gold-Bearing Formation at Open-Cast Workings in Socorro Mine.

ork is easy and cheap, and the cut goes on the strike,
the bed-rock, with a width of about 15 m. Almost
tuff pays for milling; the richest part is an ocher-
erial in the quartz vein. The bed-rock is a little
" in its dip, and on the "steps" are always found rich
As in the Concordia, all the gold is loose in the gangue.
Transval Mine.—This mine is located in the ridge
Concordia and Socorro. It is only a prospect, fitted
the beginning with an Allis-Chalmers five-stamp

mill of the type having 850-lb. stamps. The idea of the promoters of the enterprise was, that if the two adjacent properties, Socorro and Concordia, were so good, the land between them should be also very rich. Experience has not yet corroborated this presumption, and, though well equipped, the mine is not in paying condition.

The system of veins in the Transval is well defined and parallel with the Socorro system; but the ore is very low in free gold, and the pyrites are also poor, which is very remarkable in this locality.

It may be that future explorations under my direction will prove the existence of rich ore like that of the neighboring mines; but for the present only very sound metallurgy can make things right.

4. *The Madrono Mine.*—Down in the direction of Rio Cristal, the slate formation gives place to the granites at about 5 km. (3 miles) from Concordia. In this country-rock many mines have been worked, with not very favorable results. The most fully explored of them is the Madrono.

The deposit is more like a stock-work than a system of veins. The granites are altered to a great depth, not only by weathering, but also by dynamic action, and are crossed by veins and veinlets which in places reunite in knots, forming great bunches of ore.

The quartz is sugary in texture and the gold is adherent to the gangue. The distribution of the pay-shoots is very erratic; as in all the gold-formations here, the country-rock contains values—generally very small, but, in the vicinity of rich pockets, in paying amount.

Beautiful specimens of native gold (foliations, plates, dendrites, etc.) have been found occasionally in this mine.

With the exception of the Transval, all the mines are provided with native mills, and the ores are handled and milled in the crudest fashion. All transportation has been performed heretofore on men's backs, etc. I have now erected some cable-lines at Concordia and rail-lines in Socorro and Transval, which entirely change the transportation feature in the economy of the undertakings; but the most important reforms in method remain to be effected, viz.:

e transportation of the *mantos* (earthy ores) by means
in sluices or, better, in pipes.

liminary washing in trommels (fine screening) of the
immediate recovery of the fine loose gold on plates.

draulic exploitation of a great part of the deposits

ncentration of the trommel- and mill-tailings and final
of the concentrates-values by cyaniding or by exporta-
he high-grade concentrates.

IV. Conclusions.

observations are suggested by the foregoing state-

s a rule, the big quartz lodes are almost barren (say $2
r ton) in free gold.

ery frequently the big veins are accompanied by feeders
ere *aqujas*), which are generally rich.

e richest parts of these deposits are the narrow, not
ned, veinlets, much like lenticular masses.

er NE. of the group of mines I have hastily described,
spects are attracting considerable attention; small
ills have been erected; and the exploitation of rich
giving fair results.

iscovery of permanent gold-bearing formations, the
of the outcrops, and principally the *mantos* with their
inary facilities for cheap working, warrant the expecta-
t a very important future is reserved for these metal-
zones of southern Colombia.

ACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS.]

urrence of Silver-, Copper-, and Lead-Ores at the Rica Mine, Sierra Mojada, Coahuila, Mexico.

BY FRANK R. VAN HORN, CLEVELAND, OHIO.

(New York Meeting, February, 1912.)

I. Introduction.

summer of 1908, R. B. Cochran, Superintendent
ompania Metalurgica Mexicana at Sierra Mojada,
resented to the Department of Geology and Miner-
ase School of Applied Science an interesting series
, copper-, and lead-ores from the Veta Rica mine.
s contained several interesting minerals which were
in a somewhat peculiar manner, so that I was im-
make a closer study of their occurrence. The inves-
has brought out several new facts concerning the
f the district and the geological occurrence of the
therefore has seemed worthy of record. Two papers
published previously about the region, one by Chism[1]
nd the other by Malcolmson[2] in 1902. These articles,
treat of the district more from the stand-point of the
ist and mining engineer, and not much attention is
o its mineralogy and economic geology.

I. Geographical Location of the District.

erra Mojada is situated in the extreme western part of
of Coahuila, near the Chihuahua boundary. It is
est via the Mexican Central railroad from El Paso,
scalon, Chihuahua, a distance of 494 miles. From
t, a branch line, the Mexican Northern railroad, runs
in a NE. direction to the mines, which are its northern
The railroad was built about 1890, practically on
f the Sierra Mojada mining-district.

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ojada, Mexico, by Richard E. Chism, *Trans.*, xv., 542 (1886-87).
rra Mojada, Coahuila, Mexico, and Its Ore-Deposits, by James W.
, *Trans.*, xxii., 100 (1901).

III. CLIMATE AND TOPOGRAPHY.

The region is an arid one, and therefore the supply of water is small. The rainy period may occur between the middle of May and the middle of July, but during some seasons there is practically no rain. Two springs are known, and productive wells may be obtained by deep borings; but it has been found more economical and satisfactory to ship water in from points beyond Escalon on the Mexican Central railroad. The timber for mining purposes is also obtained elsewhere. On account of the conditions just mentioned, the region surrounding Sierra Mojada is an uninhabited desert.

Sierra Mojada is the name of a town as well as of a range of mountains or hills. The town lies in a valley from 2 to 3 miles wide N–S. and about 12 miles long in an E–W. direction. The altitude is about 4,800 ft., and the valley is bounded on the north by hills known as the Sierra Planchada. The southern boundary consists of the Sierra Mojada mountains, which rise more than 2,000 ft. above the valley. These mountains, which possibly more properly should be called hills, rise gradually out of the valley with the exception of the last third of their height, which ends in a steep cliff of about 600 ft. This precipice is found for a distance of 3 miles in the highest portion of the Sierra, and dies down into rounded hills on the east and west.

IV. LOCAL GEOLOGY.

The rocks which outcrop in the region are limestone and conglomerates, while underground workings show the presence of a so-called breccia.

1. *Limestone.*—The rocks which form the Sierra and constitute the chief underlying mass of the district are somewhat dense grayish-blue magnesian limestones of Cretaceous age. Thin sections show a general similarity to the Palæozoic limestones of the central and eastern United States. They are often stained yellow and red by iron oxide. According to Chism, these rocks dip about 35° S., but in places they stand nearly vertical.

2. *Conglomerate.*—In the bottom of the valley, a limestone conglomerate is found which is clearly the result of the disintegration of the limestone cliffs of the Sierra. The cement of

nglomerate is chiefly calcium carbonate. In the Veta entral shaft, the rock has a thickness of 30 m. It is cally called "wash," and was called "alluvium" by ; but Malcolmson says that in at least three places this nerate has resting upon it a still younger Cretaceous ne, which, in one locality, on a *mesa* known as La , is 100 ft. thick. This proves that the valley was formed taceous time, and depressed beneath the sea during the period. The first discovery of silver-lead ore in 1878 ade, and the first claim was located, in the conglomerate Blanca *mesa*. This seems to indicate that the ores were ed in the older limestone prior to the submergence and ion of the younger limestone. These two periods of ion with the intervening long erosion-period seem to ond to the great unconformity between Lower Cretace-omanche) and Upper Cretaceous in the Southwest.

reccia.—Mine-workings show that, between the calcare-nglomerate and the older limestone, there is a reddish-ranular rock which is called locally conglomerate, but Malcolmson calls a porphyritic breccia. This rock has ness of 106 m. in the Veta Rica central shaft. I had at posal only a drill-core about 5 in. long, a thin section hich showed a somewhat clastic appearance. There is t deal of granular quartz present, which is in places rounded, and has at times been added to by a deposi-quartz having the same extinction as the original grain. of the quartz is also angular, and shows the conchoidal e. Other grains consist of badly-decomposed feldspar ently orthoclase) which is filled with dark spots, giving a oikilitic appearance, probably due to later impregnation on oxides. Several Carlsbad twins were observed. Still grains resemble the microgranitic ground-mass of vol-ocks. A few grains of a mineral resembling magnetite me decomposed ferro-magnesian mineral are present in mounts, as is also calcite. A faint reddish stain of iron s present in most of the grains with the exception of The rock had two semi-angular inclusions within the of the drill-core. One was reddish brown, and the other rplish red. Both were very fine-grained and massive, ling volcanic igneous rocks. I am unable to say how frequent these inclusions are, but if there were many, the rock as a whole would have a distinctly brecciated appearance. To my mind, the chief rock which makes up the drill-core is decidedly similar to a rhyolite tuff, and I am not absolutely certain that it could not be a very much decomposed rhyolite with inclusions. The rock is too badly decomposed, and other data are insufficient, for accurate determination; but it is deserving of further study. This rock is fairly impermeable, and wells are sunk into it through the conglomerate in order to obtain water. It rarely, if ever, outcrops at the surface, and its contact with the underlying limestone has generally a steep dip, although in places it becomes nearly horizontal. Slickensides are found near the contact in both breccia and limestone, and the limestone below the contact is considerably broken. The plane of the contact dips northward away from the mountains, and the ores are found either on the contact between limestone and breccia or in the limestone below the contact.

The formation of the present valley, as explained by Chism, is due to the erosion of an anticlinal fold. His principal reason for this conclusion seems to be, that the limestones of the Sierra Mojada dip southward 35°, while the rocks of the Sierra Panchada have a northward dip of from 25° to 30°. The underground workings show nothing of this anticline, and, at the time of Chism's visit, the breccia was not recognized. Malcolmson accepts Chism's explanation, but observes that the folding was not so simple, because in places, notably above the San José mine, the limestones were bent nearly vertical, or almost at right angles to their normal dip. He says, "it is evident that faulting and dislocation of the strata have also taken place to some extent," and contents himself with that statement, although otherwise he agrees with Chism. Emmons,[3] in a discussion of Malcolmson's paper, suggests that similar structures might also be produced by faulting. My observations all point towards the latter conclusion, and it seems more probable that the valley was formed by faulting, and that the breccia is part of a fault-block originally formed at the surface some distance away. The breccia was later covered by the conglomerate and younger Cretaceous limestone. Evipointing to this origin are the slickensides in both and limestone, and the fact that the limestone for some e from the contact was considerably fractured and later pregnated with ore. It is also thought that the limestone present the fault scarp which has suffered but little erosion ce the dislocation. The plane of contact between brec- l limestone is generally steep, although in some places it es quite flat, and also at times is curved, or "rolls," s not true of fault-planes in general. This, in my opinion, greatest objection to the theory of faulting. Some varia- dip might be accounted for by cross-faulting, which has ound at least once on the Veta Rica property, with a of 40 feet.

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[3] Discussion of Paper of James W. Malcolmson, by S. F. Emmons, *Trans.*, xxxii., 566 (1901).

e is a third theory which is not absolutely impossible. ion has been called to the fact that the breccia has rable resemblance to a decomposed rhyolite or rhyo- f. Either of these might have come to rest upon an limestone surface, which might account for a varying f the contact-plane. In this case, the valley would have rmed by erosion, but not of an anticline. Not enough ce is known by any one to warrant a satisfactory con- concerning the origin of the breccia, and, in my opinion, urate investigation of the rock in place is needed to light on its true character. This in turn will do much s explaining the origin of the valley, as well as that of s.

ngs on the Vulcano and El Acaso claims, north of the Rica, show that a rock called locally "sandstone" or zite" is encountered below the limestone at a depth of d 346 m. respectively on the two claims. Underneath ck, a "slate" is said to be found at 440 and 455 m. rocks show a slight dip to the north rather than to the and seem to have no relation to any anticlinal fold.

V. POSITION OF MINES AND CHARACTER OF ORES.

mines of the district are situated in the foothills about rd of the distance from the floor of the valley to the t of the Sierra. There are from 15 to 20 in number, uted approximately on an E–W. line for about 3 miles. ree largest producers in 1900, according to Malcolmson, were: Veta Rica, 41,018; San José, 30,285; and San Salvador, 28,783 metric tons.

The general character of ores and their composition was given by the same author as follows:

	Metric Tons.	Ag. Oz. Per Ton.	Pb. Per Cent.	SiO_2. Per Cent.	FeO. Per Cent.	CaO. Per Cent.	$BaSO_4$. Per Cent.	Cu. Per Cent.
Basic lead-ore, .	80,000	9.0	15.0	16.0	36.0	5.0	——	——
Siliceous lime, .	52,000	18.0	0.0	20.0	3.0	21.0	10.0	0.6–3.0
Lime ore, . .	25,000	8.0	0.0	7.0	3.0	33.0	——	——
Siliceous lead-ore,	23,000	10.0	14.0	34.0	22.0	2.0	——	——

The chief ore of the district has always been a silver-lead ore consisting largely of argentiferous cerussite, but since 1893 the mines in the western part of the camp, notable among which are the Veta Rica and San José, have produced large amounts of silver-copper ores, containing little or no lead.

VI. VETA RICA ORE-BODIES.

The Veta Rica claim covers 9 hectares, or about 22.25 acres, and considerably more than $8,000,000 has been taken from the mine to date. The first shaft was sunk near the southern boundary with the adjoining Fortuna property, and was carried only to the 2d level (see Figs. 1 and 2). At the 1st level, the ore common to the camp as a whole was encountered. This was argentiferous cerussite mixed more or less with limonite, massive sulphur, and gypsum. This association is quite usual in the region, and the sulphur has been obtained in such large amounts that some shipments have been made. The gypsum especially was so constant that the miners used it as an indication of proximity to good ore. The original mineral was undoubtedly galena, and the reaction which took place was probably the following:

$$PbS + 4O = PbSO_4$$
$$PbSO_4 + CaCO_3 + 2H_2O = PbCO_3 + CaSO_4 . 2H_2O$$

There is, of course, a possibility that anhydrite may have been the original mineral, which has altered to gypsum as described by Lindgren.[4] In the present instance, since nothing but secondary minerals are found, the gypsum was probably di-

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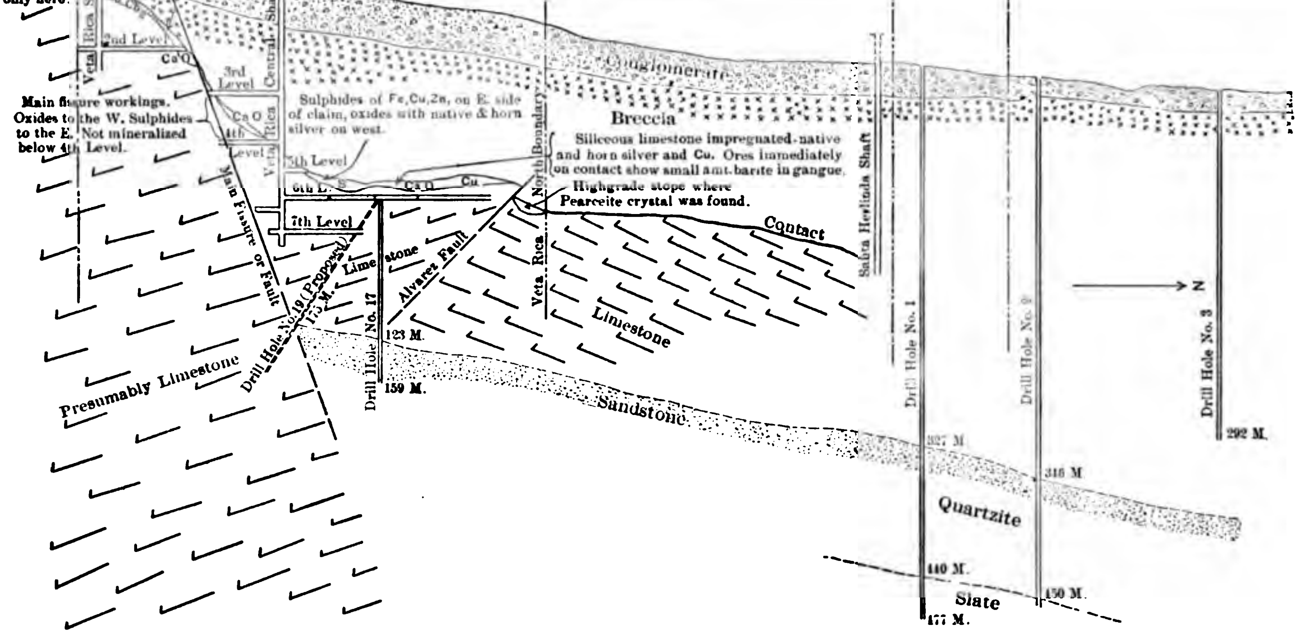
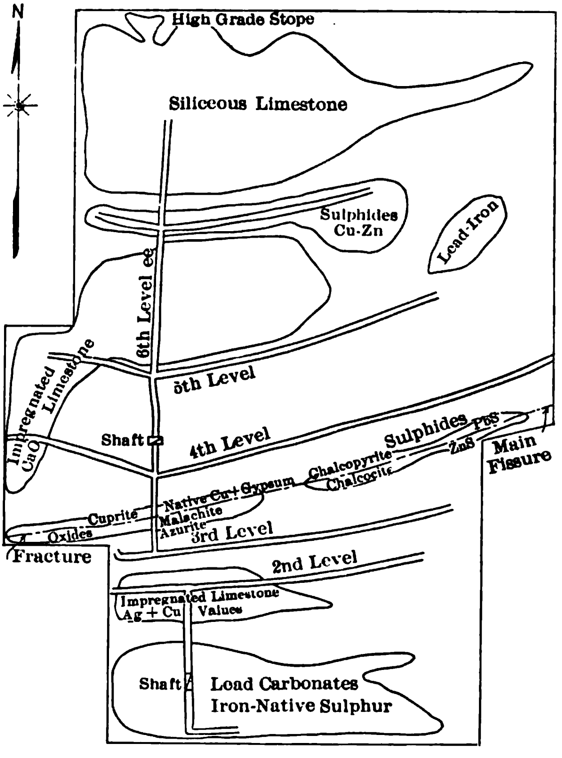
[4] W. Lindgren, Anhydrite as a Gangue Mineral, *Economic Geology*, vol. v., No. 6, p. 522 (Sept., 1910).



(Modification of drawing furnished by R. B. Cochran.)

FIG. 1.—VERTICAL SECTION IN A NORTH-SOUTH DIRECTION THROUGH VETA RICA AND ADJOINING CLAIMS.

rectly precipitated. The sulphur might have been formed by the action of hydrogen sulphide upon the limestone, or on sulphur dioxide, or by the reduction of the gypsum. The most interesting feature about this occurrence is that the gypsum and sulphur, which in other regions have undoubtedly been formed, but either dissolved or carried farther, are here precipitated. This is possibly due to the general aridity of the climate and consequent lack of large amounts of underground water. In this ore-body, the limestone foot-wall was not enriched, and the ore was very soft and required square sets in the stopes.



(Sketched by R. B. Cochran.)

FIG. 2.—GROUND-PLAN OF VETA RICA ORE-BODIES.

On the 2d level of the south shaft, a drift north through the limestone to the breccia-contact found a body of ore which consisted of a siliceous limestone impregnated with native silver, cerargyrite (var. embolite), and copper-values up to 3 per cent. At the extreme northern part of this deposit, the ore gave a fracture was discovered, which is locally called the sure" (see Fig. 2). In order to reach this more easily, to prospect the northward-dipping contact, the Veta ral shaft was sunk. At the 3d level, a drift was run th until both contact and fissure were encountered. discovered at both places, and that found along the as of the same general nature as the ore mined on the of the south shaft, but was of poorer grade. It conown to the 4th level, slightly below which the shaft gh the contact of breccia and limestone.

e-body on the "main fissure" was situated about 20 m. contact and was found only between the 3d and 4th low which mineralization did not occur. This "fisa break in the limestone nearly at right angles to the of dip of the contact-plane, but seems not to have the latter to any appreciable extent. The plane of e also dips northward, but at a much steeper angle breccia-limestone contact. The ores on this fissure r in copper than any yet mentioned, and are also rous. There are two such silver-copper shoots at this but one is composed of sulphides, while the other is (see Fig. 2). These bodies are separated by 120 m. limestone, although both are on the fissure. The ores are found on the eastern side of the claim and f chalcocite and chalcopyrite, with small amounts of chiefly as tarnish. Some galena and sphalerite are to occur here. The chalcocite is all of the gray, masety and not the sooty kind which is recognized as sec- the mines at Butte and elsewhere. On the other fact that the fissure intersects the breccia-limestone nd that mineralization extends only to the 4th level indicate that the deposit was made by later, descendions, and resulted from secondary precipitation.

idized body is on the west side of the claim and conative copper, cuprite, azurite, and malachite, all mixed less with gypsum. It is very evident that the soluch caused the oxidation of this body of ore were either of their oxidizing power before reaching the sulphide the east or were entirely prevented from contact with gh both are on the same fracture. The officials of the mine believe that a water-channel followed down the fissure on the west side, and was prevented from reaching the sulphides by the wall of barren limestone 120 m. thick. As was the case in the lead-deposits of the south shaft, the gypsum is a prominent and interesting feature of the oxide shoot. It was probably formed in the decomposition of copper sulphate according to the following reaction:

$$2CuSO_4 + 2H_2Ca(CO_3)_2 + 3H_2O = \underset{\text{(Malachite)}}{CuCO_3 \cdot Cu(OH)_2} + \underset{\text{(Gypsum)}}{2(CaSO_4 \cdot 2H_2O)} + 3CO_2$$

For the formation of azurite, the following reaction may have occurred:

$$3CuSO_4 + 3H_2Ca(CO_3)_2 + 4H_2O = \underset{\text{(Azurite)}}{2CuCO_3 \cdot Cu(OH)_2} + \underset{\text{(Gypsum)}}{3(CaSO_4 \cdot 2H_2O)} + 4CO_2$$

The cuprite and copper might have been formed by the reducing action of ferrous sulphate, after the manner suggested by Ries,[5] as follows:

$$2CuSO_4 + 2FeSO_4 + H_2O = \underset{\text{(Cuprite)}}{Cu_2O} + Fe_2(SO_4)_3 + H_2SO_4$$

$$Cu_2O + H_2SO_4 = Cu + CuSO_4 + H_2O$$

Since, however, the copper is found with gypsum like malachite, I see no reason why the same reaction which formed malachite and gypsum should not also form copper and gypsum in the presence of some reducing-agent. Any superfluous oxygen might then oxidize some of the copper to cuprite. Some of the specimens bear out this conclusion, as the cuprite is seen in small amounts on the exterior of copper crystals. In still other cases, however, it is mixed with malachite and gypsum. Of course it is also possible that both cuprite and copper might be formed by the reduction of malachite, but the copper, especially, occurs in well-defined cubes imbedded in gypsum, and could not possibly be a pseudomorph formed by the alteration of malachite.

On the north side of the central shaft, between the 5th and 6th levels, two of the largest ore-bodies in the mine were discovered. Here the contact between breccia and limestone becomes perceptibly flatter, and in this respect does not conform l fault-planes. Of course it is not absolutely certain ault does exist between the two rocks, although many int to that conclusion. The ore-body nearest the shaft es that on the "main fissure" between the 3d and 4th n that it consists of a sulphide and an oxidized portion. ve, the sulphides are on the east and consist of copper-, nd zinc-minerals, while on the west the ores are oxides bonates of copper with native silver and cerargyrite ıbolite). I saw no specimens from this ore-body, but it noted that the two sulphide bodies are both on the d that on the west side of the mine the same horizons oxides. These two bodies, with one other small occurre the only sulphides found in the mine.

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[5] H. Ries, *Economic Geology*, 3d ed., p. 339 (1910).

ıer northward, but also between the 5th and 6th levels, rge body of the type forming the largest tonnage of the "siliceous silver-lime" with small amounts of copper. ıposition of this ore ranges around the following values: oz.; Cu, 0.6 to 3 per cent.; CaO, 20; SiO_2, 22; FeO, d barite up to 10 per cent. The ore occurs in the limeelow the breccia-contact. Although there was ore on tact, it was not so good as below, and was said to con' an argentiferous earthy gangue containing considerite." The specimens of the chief ore below the conich were seen by me consisted of red granular limeonsiderably fractured. Many of the pieces showed surfaces resembling rough slickensides. Under the ope, the rock consists of coarser grains of calcite, mixed er ones which seem to be the result of grinding, and uartz. The cracks between the larger particles are ith embolite and some silver. There is no indication eplacement, and the ore must be considered as an imion of fractured limestone. The solutions evidently ed some iron, which has produced the reddish stain. h the ore is said to contain workable amounts of copcopper-minerals were recognized.

06, during the progress of work in the northern portion silver-lime" very near the north boundary of the claim, was encountered. This is known as the Alvarez fault . 1), and has a throw of 40 ft. The fault-plane dips to the south, and must intersect the "main fissure" at no great depth below the present workings. After working down to the limestone-contact, it was found that the character of the ores had completely changed, and rich silver-copper ores were found in fractured grayish-blue limestone. In places there is a great deal of barite present. This was known as the "high-grade stope;" the ores often ran up to 150 oz. of silver per ton, with some copper. The minerals found here were native silver, argentite, proustite, pearceite, and erythrite. The pearceite, $(Ag, Cu)_{16}As_2S_{11}$, which is arsenical polybasite, has been found previously at but four other localities in the world: Schemnitz, Hungary; Arqueros, Chile; Aspen, Colo.; and Marysville, Mont. The Veta Rica pearceite contained 15.65 per cent. of copper, but no antimony. Tests on the argentite and proustite show that both contain copper, and indicate that the solutions which deposited this ore contained chiefly silver, copper, arsenic, and sulphur. Thin sections of one of the specimens show a mass of lamellar, plumose barite impregnated with argentite, and silver. In most places the silver surrounds the argentite, and was probably formed by the reduction of the latter. In a few instances, small grains of silver were coated with a black film of argentite. If the native silver is excepted, the only common oxidation-product found here is the erythrite, which often occurs in well-defined crystals in cavities and cracks in barite. This is the first occurrence of cobalt-minerals which has been observed in Sierra Mojada. It seems probable that cobalt arsenides are present in some of the ores, but they have not been detected. Since this particular association of erythrite and barite was said to contain more than 100 oz. of silver per ton, the erythrite was carefully examined and was found to be coated in places with exceedingly fine hairs of native silver, which sometimes accumulated into globular masses. In all cases, the silver was deposited on the erythrite and was therefore formed later. I have no means of ascertaining what relation this association had to the occurrence of silver with argentite, proustite, and pearceite. However, it would seem that in both associations native silver was one of the most recent minerals formed.

VII. PARAGENESIS AND ORIGIN OF THE ORES.

In the description of the various ore-bodies, the association of the different minerals has been given, and where possible,

has been called to the order of their deposition.
wever, most of the specimens at my disposal are in
pieces, and nothing is known of their relations to each
definite conclusion can be reached on the order of
n of the various minerals. This can only be deter-
the mine itself. There a distinct association seems
own between silver and copper on one hand, and be-
lver and lead on the other. This would conform to
mediate solubility of silver sulphate when compared
d and copper sulphates. The association of lead and
s seen on the 1st level of the south shaft, and is ap-
the most common one in the district. It was likewise
one known from 1878 until 1893, when the first copper-
discovered. Since most of the ores are oxidized, very
is found with them—in accordance with the greater
of zinc sulphate as compared with the sulphates of
ilver, and lead. In one or two mines a little earthy
onate has been found below lead carbonate ores in the
below the contact. In several mines, lead carbonate
aining silver was found below the copper-silver ores.
especially true in the San José mine, and seems to
that a migration of solutions had occurred in which
silver ores had wandered farther than those containing
lver—a phenomenon which does not agree with the
olubility of their sulphates. This might be explained
resence of certain compounds which would render the
e soluble, or by the assumption that two different solu-
osited the ores at two different periods.

ard to the origin of the ores, the data at my disposal
sufficient to warrant any conclusion. It is, however,
r that there is a definite relation between the deposi-
re, and the plane of contact between breccia and lime-
hether this be a fault-plane, or an ore-channel between
milar rocks. All ores are found below the breccia,
considered quite impervious, and therefore would
uch affected by either ascending or descending solu-
Since most of the ores are oxidation-products, and
secondary, the question of origin evidently depends
interpretation of the three sulphide bodies found in
e. These are either parts of an original deposition,

along the contact of limestone and breccia, and in the "main fissure," which has resisted oxidation, or they have been precipitated after secondary concentration. The various minerals found in the sulphide ores, such as chalcocite, chalcopyrite, argentite, prousite, pearceite (polybasite), and silver, according to the writings of Lindgren[6] and W. H. Emmons,[7] are possible in primary deposits as well as secondary enrichments. Neither are there in the Veta Rica deposits any convincing criteria of downward enrichment as suggested by Ransome.[8] According to Sales,[9] gray massive chalcocite, such as occurs here, is strongly indicative of primary origin. The presence of sulphide ores in the "main fissure," about 20 m. from the contact of breccia and limestone, and the fact that they are found only between the 3d and 4th levels, rather indicate descending solutions. In regard to the "silver-lime" ores which are found below the contact, as impregnations of cracks in the limestone, the indications are that the deposits were probably formed by descending waters which had leached the material from the contact-plane above. The presence of considerable cerargyrite (var. embolite) seems to be rather characteristic of arid regions,[10] and is believed by Keyes[11] to have been largely due to wind-blown saline dust, taken up by meteoric water, which causes the alteration of ore-bodies, with the consequent formation of silver chloride. If, however, the observation is correct that the Sierra Mojada ores were formed before the deposition of the conglomerate and the younger Cretaceous limestone, it is evident that the action of percolating sea-water would have tended to cause the formation of silver chloride long before the present arid conditions prevailed. The presence of small amounts of

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[6] W. Lindgren, The Relation of Ore-Deposition to Physical Conditions, *Economic Geology*, vol. ii., No. 2, p. 105 (Mar.-Apr., 1907).

[7] W. H. Emmons, A Genetic Classification of Minerals, *Economic Geology*, vol. iii., No. 7, p. 611 (Oct.-Nov., 1908).

[8] F. L. Ransome, Criteria of Downward Sulphide Enrichment, *Economic Geology*, vol. v., No. 3, p. 205 (Apr., 1910).

[9] R. H. Sales, Discussion of Criteria of Downward Sulphide Enrichment, *Economic Geology*, vol. v., No. 7, p. 681 (Oct.-Nov., 1910).

[10] R. A. F. Penrose, Jr., Superficial Alteration of Ore-Deposits, *Journal of Geology*, vol. ii., No. 3, p. 314 (Apr.-May, 1894).

[11] C. R. Keyes, Cerargyritic Ores: Their Genesis and Geology, *Economic Geology*, vol. ii., No. 8, p. 774 (Dec., 1907).

e in the silver chloride might be accounted for in this
r.

VIII. Conclusion.

nkly admit that I have been unable to arrive at any con-
; conclusion regarding certain geological features pre-
in this district. Most evidence points to the fact that
ley was made by faulting and not by the erosion of an
ie, and that the ores were deposited in a fault-plane or
nnel between breccia and limestone. All this occurred
the deposition of the younger Cretaceous limestone.
iost of the ores are now oxidized, it seems most probable
e few sulphide bodies are portions of the original depo-
and have not been formed by secondary concentration.
st that I have been able to state a few facts which were
eviously known about a region which is of import eco-
lly, and interesting geologically. It is to be hoped that
e who may visit the district will pay especial attention to
ious features which are still open to discussion, such as
mation of the valley and the origin of the breccia.

REVISION.

ACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS.]

Vanadium in Pig-Iron.

BY PORTER W. SHIMER, EASTON, PA.

(New York Meeting, February, 1912.)

s not seem to be generally known that some American contain notable amounts of vanadium, and while the investigation is far from covering the whole range of is hoped that, at least, it may serve to call attention to ortance of looking for this element. The majority of rons investigated were made in the Lehigh Valley and ing furnaces, using more or less New Jersey magnetite mixtures. Many of these ores, and probably all of ntain a little vanadium as well as titanium; the ore of the largest of these mines contains 0.05 per cent. ium and 0.60 per cent. of titanium.

iomas M. Drown was the first to detect the presence ium in the incrustations[1] occurring on certain pig-irons, lso found this element in a number of magnetites, but iods for its accurate determination had not, at that n well worked out. In a general way, it may be said Lehigh Valley pig-irons contain from 0.02 to 0.05 per vanadium and from 0.10 to 0.20 per cent. of titanium. elements follow the silicon; that is, the furnace-con- which produce a high-silicon, low-sulphur iron will maximum reduction of vanadium and titanium, and r grades of iron, with low silicon and high sulphur, ower content of these elements. The following an- irons made from the same mixture, but under varying conditions, show this very well:

	Vanadium. Per Cent.	Silicon. Per Cent.	Sulphur. Per Cent.
). 1 X, . . .	0.042	3.46	0.022
). 2 plain, . .	0.038	2.26	0.041
ottled, . . .	0.029	0.52	0.106

---

on and Firmstone, Note Concerning Certain Incrustations on Pig Iron, , 641 (1883–84).

The following are the results of vanadium-determinations in a number of pig-irons made in the Lehigh valley and in New Jersey:

	Vanadium. Per Cent.
Sample A.	0.028
Sample B,	0.032
Sample C,	0.039
Sample D,	0.050
Sample E,	0.049
Sample F,	0.046

That not all pig-irons made wholly of magnetic ores contain these notable amounts of vanadium is shown by the fact that two widely-separated furnaces, using all magnetite, none of which is from New Jersey, were found to contain respectively 0.006 and 0.012 per cent. of vanadium. An iron made wholly of Virginia hematite ore contains 0.014 per cent. of vanadium, and two different irons, presumably made wholly from Lake Superior ores, contain respectively 0.009 and 0.023 per cent. Vanadium sometimes occurs in limestone and in the ash of coke, and these no doubt may contribute a part of the vanadium which finds its way into the iron.

Determinations of vanadium in iron-castings made at different times and places in the Lehigh valley showed vanadium-contents as follows: 0.033, 0.013, 0.022, 0.024, and 0.027 per cent. To ascertain the loss of this element on remelting in the cupola, a mixture of three irons was used in which the vanadium had been carefully determined just before charging. The average content of these irons was 0.0313 per cent. and the average content of the castings was 0.027 per cent., showing a loss of 13.74 per cent. of vanadium on remelting.

There is reason to believe that the effect of vanadium on the strength and other qualities of these irons is noticeably beneficial, and titanium probably acts in a similar way. When these elements are present in notable amounts there is often a peculiar fine-grained structure combined with softness and unusual strength. I have often seen pig-iron made wholly or in large part of New Jersey magnetites which showed this grain to a very marked degree. Such irons often present a peculiar mottled appearance well described by Bauerman in his description of titanium pig-iron as made up of "dull, dark gray patches set in a white net work." Superficially, the appearance f hard, mottled iron, but, in reality, the iron is very a when the silicon is well below 1 per cent. The sul- o, is quite low in these irons. Sometimes the fracture iron will show partly normal gray iron and partly the mottle; the lower half of the pig may be mottled and er half gray, or there may be simply a mottled spot led by gray iron. It may be noted, however, that, exception, the fine-grained mottled part of the pig is iceably softer than the gray part of the same pig. The are curled and partake of a steely character. This finishes beautifully in the lathe, and takes a high

long been my view that the peculiar grain is due to n or titanium, or to both together. The latter ele- esent in the iron as a carbide, is often as high as 0.30 . Neither vanadium nor titanium seems to be alloyed iron; both are probably present as carbides; at all hey are found almost wholly, if not entirely, in the in- graphitic residue left after the iron has been dissolved dilute hydrochloric acid. The vanadium compound is en finer state of division than the titanium carbide, atter occurs in cubical crystals that can be readily d from the insoluble residue. (See my paper on a Carbide in Pig-Iron.[2])

neither vanadium nor titanium is alloyed with the nay be asked how they can have any effect on its struc- trength. Apart from any cleansing action these ele- nay have by removing any nitrogen and oxygen that present, the suggestion is here ventured that they e structure by separating out earliest during the solidi- of the iron as uniformly-distributed, finely-divided car- hich furnish starting-points or nuclei for the rapid zation of the iron, in this way causing a finer grain— accompanying greater strength—than would be the he absence of these elements. That titanium carbide stallizes out while the iron is still soft is proved by its harp-edged cubical crystals.

s connection, the results of a test of a vanadium-steel, containing 0.143 per cent. of vanadium, may be of interest. After dissolving a sample of the steel in dilute hydrochloric acid (25 cc. of HCl, 1.20 sp. gr., to 200 cc. of water), the black, carbonaceous residue was filtered off and in it was found 0.068 per cent., or nearly one half, of the total vanadium. On dissolving the steel in a cold solution of double chloride of copper and potassium the residue again contained 0.068 per cent. of vanadium. Thus, even in the standard vanadium-steel, about one-half of this element is present in an unalloyed form, probably as a carbide.

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[2] *Trans.*, xv., 455 (1886–87).

As to the addition of vanadium to pig-iron for the purpose of increasing its strength and refining its grain, it is probably safe to assume that greater improvement may be looked for when it is added to weak irons which do not already contain notable amounts of this element.

) REVISION.

ACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS.]

Slime-Filtration.

of the paper of George J. Young, presented at the San Francisco meet-
ctober, 1911, and published in *Bulletin* No. 59, November, 1911, pp.
872.

M. Nicholas, Melbourne, Australia (communication
cretary): * Professor Young deals with quite a variety
ines and methods, but, strangely enough, makes no
of the work done by me, although he concludes with
rk that the "thin-cake continuous filters are a decided
nent over the thick-cake filters."

se interested in slime-treatment may consider that I did
more than to patent a process, I submit Fig. 1, showing
ing-drawings of a machine, made, tested, and proved
satisfactorily by me early in 1905. This time is prior
ork done in slime-filtration with rotating filters by E.
r or others. I maintain that a number of appliances
se for slime-filtration infringe my patents. And after
austive inquiry, I know of no machine or process, prior
scription in 1899,[1] that embodied getting rid of the
by holding them on the cloth with the vacuum, and
harging them with back-pressure, so as to form a cycle
peration, the result being continuous treatment. I
think that any paper dealing exhaustively with this
s not complete without a reference to the work that I
e.

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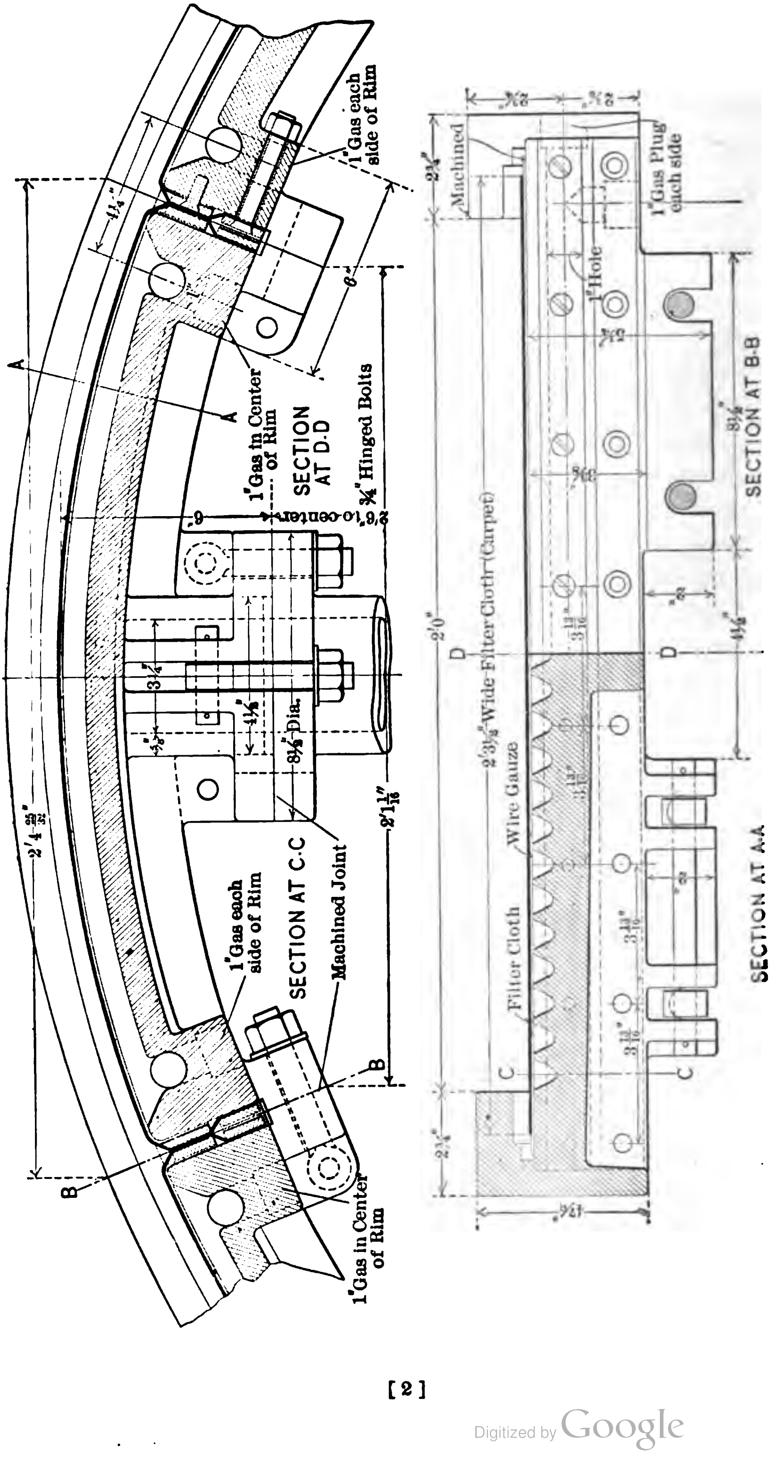
* Received Mar. 11, 1912.

[1] U. S. Patent No. 619,211, Feb. 7, 1899.



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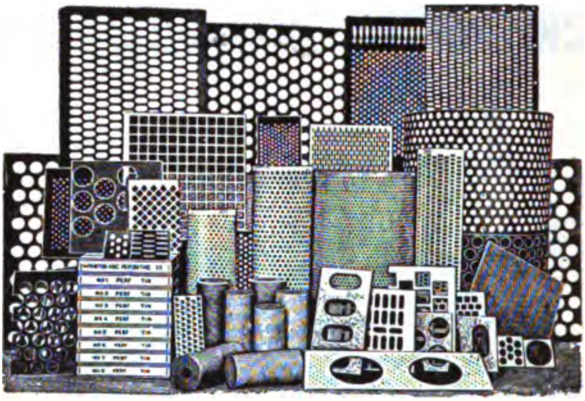

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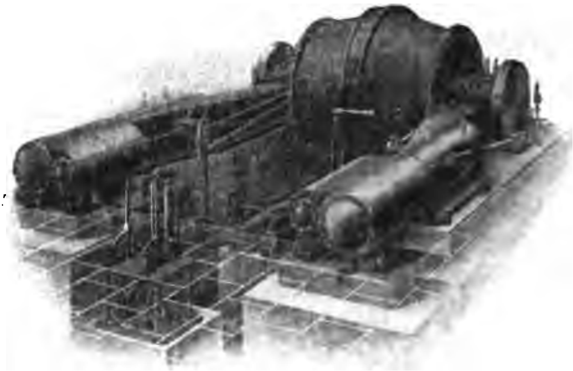

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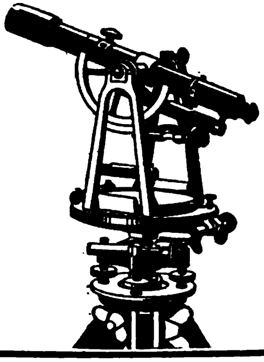

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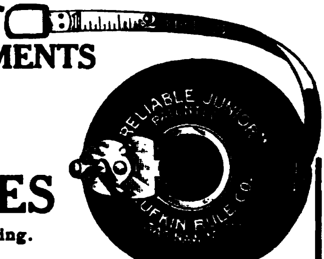

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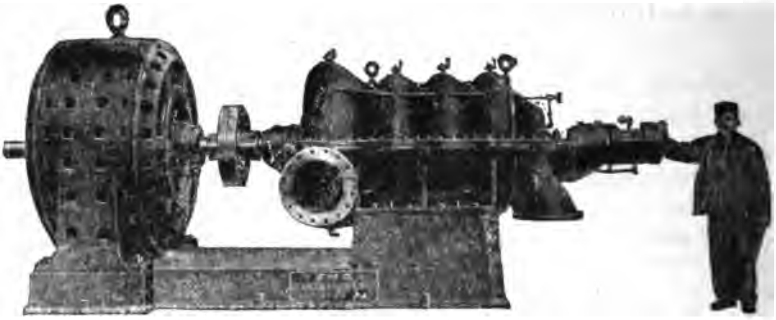

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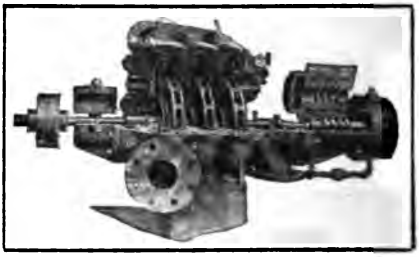

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CONTENTS:

SECTION I.—INSTITUTE ANNOUNCEMENTS.

PAGE
f Officers and Committees, ... ii
ite Announcements, ... v
y, ... xii
ership. xvii

SECTION II.—PAPERS AND DISCUSSIONS.

ecomposition of Metallic Sulphates at Elevated Temperatures in
Current of Dry Air. By H. O. Hofman and W. Wanjukow, . 889
less Combustion (and Discussion). By Carleton Ellis, . . 945
c Hoist at the Hecla Mine. By E. M. Murphy, . . . 977
ncy-Engineering Applied to Mining. By G. A. Collins, . . 995
g-Methods at Nocozari, Sonora, Mexico. By D. C. Livingston, . 1009
ceable Lips for Elevator-Buckets. By H. J. Maguire, . . 1017

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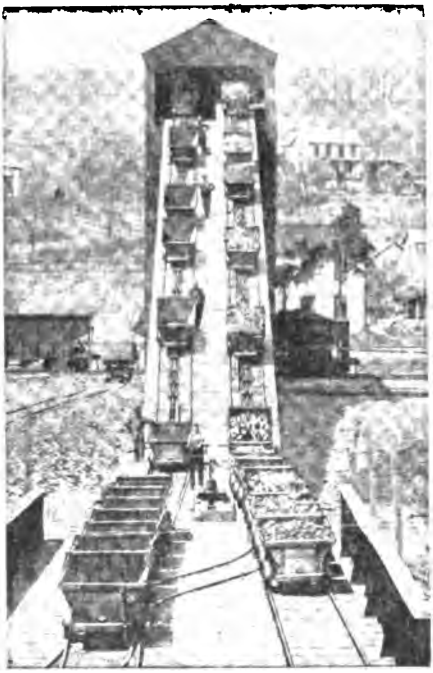

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TABLE OF CONTENTS.

SECTION I. INSTITUTE ANNOUNCEMENTS.

PAGE
fficers and Committees for the Year Ending February, 1913, . ii
Announcements, ... v
. . . . xii
hip, ... xvii

SECTION II. TECHNICAL PAPERS.

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Electric Hoist at the Hecla Mine. By E. M. MURPHY, . . . 977
Efficiency-Engineering Applied to Mining. By G. A. COLLINS, . 995
Mining-Methods at Nocozari, Sonora, Mexico. By D. C. LIVINGSTON, ... 1009
Replaceable Lips for Elevator-Buckets. By H. J. MAGUIRE, . 1017

pers contained in Section II. have been so printed and arranged (blank
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For the year ending February, 1913.

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Membership:—Joseph Struthers, Karl Eilers, Sidney J. Jennings, Benjamin B. hayer, John H. Janeway, Jr.

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Iron and Steel Division:—Charles Kirchhoff, *Chairman*; Charles F. Rand, *Vice-hairman*; Bradley Stoughton, *Secretary*; John Birkinbine, James Gayley, Henry . Hibbard, Henry M. Howe, Robert W. Hunt, Julian Kennedy, Charles K. eith, Richard Moldenke, Joseph W. Richards, Albert Sauveur, F. W. C. hniewind, A. A. Stevenson, Felix A. Vogel, Leonard Waldo, William R. ebster.

CORPORATION.

Finance:—Charles F. Rand, James Gayley, George C. Stone.

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INSTITUTE REPRESENTATIVES.

United Engineering Society Trustees:—Joseph Struthers (1913), Theodore Dwight 914), James F. Kemp (1915).

Joint Conference Committee of the Four National Engineering Societies:—James . Kemp, Joseph Struthers.

John Fritz Medal Board of Award:—E. Gybbon Spilsbury (1913), R. V. Norris 914), Charles Kirchhoff (1915), J. F. Kemp (1916).

American Association for the Advancement of Science:—H. O. Hofman, John D. ving.

Committee No. 24, International Association for Testing Materials:—Henry D. Hib- rd.

International Engineering Congress, San Francisco, 1915:—Samuel B. Christy, illiam C. Ralston, Edwin T. Blake.

Kelvin Memorial Committee:—James F. Kemp, Samuel B. Christy, James Doug-
s, Joseph Struthers.

Affiliated Student Societies.

Any society of undergraduates at a technical school, comprising students in any branch of engineering, metallurgy, chemistry, geology, etc., may be recognized by the Council in its discretion as an Affiliated Student Society. A circular giving details of the plan of affiliation may be obtained on application to the office of the Secretary of the Institute.

The following societies have been placed by authority of the Council on the above list:

AFFILIATED STUDENT SOCIETIES.

The Mining Society of the Sheffield Scientific School, Yale University, New Haven, Conn. *President*, Karl C. Stadtmiller; *Secretary*, S. B. Gordy.

The University of Illinois Student Branch of the American Institute of Mining Engineers, Champaign, Ill. *President*, Leonard V. Newton; *Secretary*, L. W. Swett.

The Engineering Society of the University of Nevada, Reno, Nev. *President*, D. E. Bruce; *Secretary*, R. M. Seaton.

The University of Wisconsin Mining Club, Madison, Wis. *President*, Rudolph J. Stengl; *Secretary*, Mack C. Lake.

The Mining and Geological Society of Lehigh University, South Bethlehem, Pa. *President*, Willis B. Clemmitt; *Secretary*, W. Clifford Rehfuss.

The School of Mines Society of the University of Minnesota, Minneapolis, Minn. *President*, Emory P. Baker.

The Mining Engineering Society of the Massachusetts Institute of Technology. *President*, L. B. Duke; *Secretary*, Lionel H. Lehmaier.

The Student Auxiliary Society of the American Institute of Mining Engineers of the University of Kansas, Lawrence, Kan. *President*, A. H. Mangelsdorf; *Secretary*, C. J. Hainbach.

The Associated Miners of the University of Idaho, Moscow, Idaho. *President*, James W. Gwinn; *Secretary*, J. Wallace Strohecker.

The State College of Washington Mining and Geological Society, Pullman, Wash. *President*, H. E. Doelle; *Secretary*, B. R. Kinney.

The Tejas Technical Society, School of Mines, University of Texas. *President*, G. C. Cartwright; *Secretary*, David S. Alley.

The Ohio State University Student Branch of the American Institute of Mining Engineers, Columbus, Ohio. *President*, Hugh B. Lee; *Secretary*, E. P. Elliott.

The Stanford Geology and Mining Society, Stanford University, Cal. *President*, B. E. Parsons; *Secretary*, E. D. Nolan.

The Senior Mining Society of Columbia University, New York, N. Y. *President*, Roger L. Strobel; *Secretary*, Clark G. Mitchell.

Mining Association of the University of California, Berkeley, Cal. *President*, Frank L. Wilson; *Secretary*, Stanley L. Arnot.

Tufts College Chemical Society, Tufts College, Mass. *President*, P. G. Savage; *Secretary*, W. S. Frost.

University of Washington Mining Society, Seattle, Wash. *President*, Horace H. Crary; *Secretary*, Clinton R. Lewis.

Student Branch of the American Institute of Mining Engineers, Iowa State College, Ames, Iowa. *President*, M. B. Hadley; *Secretary*, R. L. Hurst.

Missouri Mining Association of the Missouri School of Mines, Rolla, Mo. *President*, D. L. Forrester; *Secretary*, J. S. Irwin.

The Pick and Shovel Club of the Case School of Applied Science, Cleveland, Ohio. *President*, L. B. Riddle; *Secretary*, S. C. Stillwagon.

Colorado School of Mines Scientific Society, Golden, Colo. *President*, Alan Kissock; *Secretary*, George Wilfley.

Mining Engineering Society of the University of Arizona, Tucson, Ariz. *President*, James J. Flanigan; *Secretary*, H. O. Coles.

INSTITUTE ANNOUNCEMENTS.

Special Notice.

The *Bulletin* is now entered at the Post Office at Second-Class Postage rate of one cent per pound, and in order to preserve this privilege it is necessary that the dues of members be paid within four months of January 1 of each year. If the dues are not paid within the period mentioned, a member's name must be removed from the regular subscription-list and the *Bulletin* mailed at the transient second-class postage rate of one cent for each four ounces or fraction thereof, prepaid by stamps affixed. It is therefore earnestly requested that dues be paid promptly—otherwise the Institute will be put to additional expense of postage and to added labor in removing and replacing names from the regular list, and in maintaining an additional separate mailing-list.

Adjourned Annual Business Meeting of the Institute.

At the Annual Business Meeting of the Institute, Feb. 20, 1912, it was voted to postpone the consideration of the proposed amendments to the Constitution until June 3, 1912, and the meeting adjourned to reconvene on that date, or on such subsequent date as should be fixed by the Special Committee of Five appointed on Feb. 20, and the Board of Directors of the Institute, acting jointly.

Due to various causes, including absence of two members of the Committee in Europe for two months, it has not been possible for the Special Committee of Five to complete its report in time for preliminary distribution to members of the Institute, and consideration by them prior to June 3, 1912, and it has been mutually agreed by the Special Committee and the Board of Directors to postpone the date at which the adjourned meeting shall be reconvened from June 3, 1912, to Oct. 7, 1912.

Nominating Committee.

At the meeting of the Council, June 28, 1912, the President announced the appointment of the following Nominating Committee to present names for officers of the Institute to be voted upon at the Annual Business Meeting, Feb. 18, 1913.

Philip N. Moore, St. Louis, Mo., *Chairman.*
John Birkinbine, Philadelphia, Pa.
David W. Brunton, Denver, Colo.
Seeley W. Mudd, Los Angeles, Cal.
Charles P. Perin, New York, N. Y.
Arthur L. Walker, New York, N. Y.

Iron and Steel Division.

COMMITTEE.

CHARLES KIRCHHOFF, *Chairman.*
CHARLES F. RAND, *Vice-Chairman.*
BRADLEY STOUGHTON, *Secretary*, 165 Broadway, New York.

John Birkinbine,
James Gayley,
Henry D. Hibbard,
Henry M. Howe,
Charles K. Leith,
Robert W. Hunt,
Julian Kennedy,
Richard Moldenke,
Joseph W. Richards,
Albert Sauveur,
F. W. C. Schniewind,
A. A. Stevenson,
Felix A. Vogel,
Leonard Waldo,
William R. Webster.

Of the iron and steel papers to be presented at the coming meeting of the Institute, to be held in Cleveland, Ohio, the latter part of October, the following are now printed, and will soon be sent to all members known to be interested in iron and steel, and to any member of the Institute applying for them:

The Effect of Alumina in Blast-Furnace Slags, by J. E. Johnson, Jr., Ashland, Wis.

A Novelty in Open-Hearth Steel-Furnace Practice, by N. E. Maccallum, Phœnixville, Pa.

Notes on Ruff's Carbon-Iron Equilibrium Diagram, by Henry M. Howe, New York, N. Y.

There are being printed the following papers:

The Case-Hardening of Special Steels, by Albert Sauveur and G. A. Reinhardt, Cambridge, Mass.

The Utility of Efficiency-Records in the Manufacture of Iron, by John J. Porter, Staunton, Va.

Notes on Titanium, and the Cleansing-Effect of Titanium on Cast-Iron, by Bradley Stoughton, New York, N. Y.

Notes on the Structure of Steel, by William Campbell, New York, N. Y.

The Manufacture of Coke, by W. H. Blauvelt, Syracuse, N. Y.

Method of Surveying and Sampling Diamond-Drill Holes, by E. E. White, Ishpeming, Mich.

A Comparison of the Action of Various Carbonizing Materials, by Robert R. Abbott, Cleveland, Ohio.

An Investigation into the Relations Existing Between Hardness as Measured by the Brinell and Scleroscope Methods and Depth of Carbonization upon Different Kinds of Steel Hardened under Different Conditions, by Mark A. Ammon, Cleveland, Ohio.

Among the papers promised for the Cleveland meeting are the following:

Blowing-In of Blast-Furnaces, by Ralph H. Sweetser, Columbus, Ohio.

On Blast-Furnace Air-Pressures, by J. E. Johnson, Jr., Ashland, Wis.

Cuyuna Iron-Ore Range, by Walter A. Barrows, Jr., Duluth, Minn., and Carl Zapffe, Brainerd, Minn.

The Manufacture of Coke, by F. E. Lucas, Sydney, C. B.

The Heat-Treatment of Steel-Castings, by C. D. Young, Rolla, Mo.

N. M. Wittorff's Preliminary Investigation of Primary Crystallization and Subsequent Physico-Chemical Permutations in Iron-Carbon Alloys Containing More than 4 Per Cent. of Carbon, by Bradley Stoughton and Simon Marcus, New York, N. Y.

Cleveland Meeting.

Preliminary Announcement.

The 103d meeting of the Institute for the presentation and discussion of technical and professional papers will be held in Cleveland, Ohio, during the latter part of October, 1912, probably the 29th, 30th, and 31st. Details of the time and place of the technical sessions, committees, and other matters of interest, will be published later. Special attention is to be devoted to subjects relating to the manufacture of iron and steel, but, in addition to the list of papers given on p. vi, there will also be presented the following:

Centrifugal Machines as Applied to Ore-Grading and Ore-Concentrating, by Godfrey T. Vivian, Cornwall, England.

The St. Helen's Mining District, Skamania, Wash., by Horace V. Winchell, Minneapolis, Minn.

Mine-Fires in Metalliferous Mines, by George J. Young, Reno, Nev.

Present Conditions of Mining in the District of Vladivostok, Siberia, by Albert F. J. Bordeaux, Vladivostok, Siberia.

The Sampling of Gold-Bullion, by Frederick P. Dewey, Washington, D. C.

A Titaniferous Iron-Ore Deposit in Boulder County, Colo., by E. P. Jennings, Salt Lake City, Utah.

An Experience in the Use of Water-Power, by C. M. Myrick, San Francisco, Cal.

Our National Resources and our Federal Government, by R. W. Raymond, New York, N. Y.

The Ta-yeh Iron-Ore Deposits, Hu-pei Province, China, by C. M. Weld, Lowmoor, Va.

Clinton Iron-Ore Deposits in Kentucky and Tennessee, by S. Whinery, New York, N. Y.

The Wood Flotation-Process, by Henry E. Wood, Denver, Colo.

A Graphic Solution of D'Arcy's Formula for the Transmission of Compressed Air in Pipes, by Nathaniel Herz, Lead, S. D.

Other papers will doubtless be received before the meeting takes place, and it is desired that all manuscripts be sent to the Secretary of the Institute not later than Oct. 10, 1912.

New York Local Section.

The New York Local Section of the American Institute of Mining Engineers was organized Sept. 22, 1911. All members of the Institute residing or doing business within 50 miles of New York City constitute the membership of this Section.

During 1911 and 1912 the Section held ten meetings, at which the following papers were read and discussed:

Iron-Ore Mining in Cuba, by Jenning S. Cox, Jr.

Warren Copper-Mining District of Arizona, by Dr. James Douglas. (Preceded by a dinner.)

The Iron and Steel Industry of India, by Charles P. Perin.

The Conservation of Health and Life in Mining, by Dr. Frederick L. Hoffman.

Practical Safety Devices, with Particular Reference to Mining, by Dr. W. H. Tolman.

The Institute Party in Japan, by Dr. R. W. Raymond.

The South African Diamond-Fields, by Dr. Gardner F. Williams.

The Design and Mechanical Features of the California Gold-Dredge, by Robert E. Cranston. (Joint meeting with the American Society of Mechanical Engineers.)

Iron-Mines in Swedish Lapland, by Prof. James F. Kemp.

Flameless Combustion, by Carleton Ellis.

The Mineral Relations in Copper Sulphide Ores as Revealed by the Microscope, by Prof. L. C. Graton and James Murdoch.

The meetings were well attended and appreciated, as was evidenced by the increased and regular attendance. The committee decided, in February, 1912, to hold smokers in the offices of the Institute at the close of the meetings to promote social intercourse among the members.

The present committee desires to increase further the activity of the members of the Section, and has formulated the following tentative plans for the ensuing year:

Eight meetings will be held during the year. Five of these will be devoted to the reading and discussion of technical papers on mining, metallurgy, electro-metallurgy, milling, and geology. The remaining three will be given over to semi-popular lectures on mining, metallurgy, and geology. All members of the Section will be invited to attend the meetings, and ladies will be invited to attend the three special meetings. Any suggestions which members may offer to the committee will be greatly appreciated.

GEORGE F. KUNZ, *Chairman.*
E. GYBBON SPILSBURY, *Vice-Chairman.*
BENJAMIN B. LAWRENCE.
THOMAS ROBINS.
LOUIS D. HUNTOON, *Secretary,*
165 Broadway, New York, N. Y.

Spokane Local Section.

A very successful and interesting meeting of the Spokane Local Section of the Institute was held in Republic, Wash., July 26, 27, and 28, about 50 members and guests being in attendance.

The meeting convened in Woodman Hall, Republic, on the evening of July 26, and a cordial welcome was extended by the Hon. J. E. Ritter, Mayor of Republic. Prof. F. A. Thomson presided, in the absence of the Chairman, R. S. McCaffery, and Vice-Chairman, G. A. Collins. Edward C. Morse presented a paper on the San Poil Mill, which was followed by an animated discussion of the various details of practice.

On July 27, an excursion was made to Eureka Gulch, visiting the Quilp, Surprise, Lone Pine, Blacktail, Knob Hill, Ben Hur, and San Poil Mines, also the milling-plants of the San Poil and the North Washington Power & Reduction Co. An informal smoker and entertainment was held at the Republic Theater in the evening.

The morning of July 28 was devoted to a visit to the Republic mine, the Princess Republic mine, and the mill of the Rathfon Reduction Co. A delightful luncheon, accompanied with excellent music, was served at noon in Woodman Hall, 125 members, guests, and ladies participating. H. W. Newton, Superintendent and Metallurgist of the mill of the North Washington Power & Reduction Co., gave an informal talk on the mill-practice, drawing special attention to the recovery of gold and silver.

A brief business session was held in the afternoon, and the meeting concluded with a visit to the mill of the North Washington Power & Reduction Co. in the evening.

L. K. ARMSTRONG, *Secretary.*

A New Volume on Ore-Deposits.

Ore-Deposits—a continuation of the "Posepny" Volume.

Comprising Papers Descriptive of Ore-Deposits and Discussions of Their Origin, Edited, with an Introduction, by Dr. S. F. Emmons. The volume contains also a Biographical Notice of Dr. Emmons by his associate and friend, Dr. George F. Becker, and a comprehensive Bibliographical Index of the Science of Ore-Deposits, prepared by Prof. John D. Irving, of the Sheffield Scientific School of Yale University. Dr. Emmons had finished his editorial work and written his Introduction before his lamented death in 1910; and the Volume contains his last words upon the subject to which he had given the work of his life, and on which he was justly regarded as the foremost authority.

Further details concerning time of publication, price, etc., will follow.

Library Research-Work.

The attention of members of the Institute is again directed to the research-work done by the librarian and his assistants, which should attract special attention from those members who have no access to the literature of subjects in which they may be interested.

During the year 1911 there were 143 searches made for members and non-members of the Founder Societies, and copies of the references have been preserved for the use of others. This work has been largely based on requests sent in by mail, from Japan, South Africa, Mexico, Canada, and England, as well as from different parts of the United States. The Librarian is confident that if it were more widely known that the library is equipped to undertake researches, the demand would increase beyond the ability of the present force to handle it. The library receives more than 700 technical periodicals which are available through the indexes for this special purpose.

Back Volumes of the Transactions.

The Board of Directors has authorized the following offers of sets of back volumes of the *Transactions*, at considerably reduced prices, to Members, Libraries, and Scientific Societies:

	Per Set.
I. Five volumes, bound in half-morocco, from No. 36 (1906) to No. 40 (1910),	$20
II. Ten volumes, bound in half-morocco, from No. 31 (1902) to No. 40 (1910), including Mexican Volume,	35
III. Twenty volumes, bound in half-morocco, from No. 21 (1893) to No. 40 (1910),	50
IV. Thirty volumes, bound in half-morocco, from No. 11 (1883) to No. 40 (1910),	60
V. Thirty-nine volumes, bound in half-morocco, from No. 1 (1873) to No. 40 (1910), with the exception of No. 10 (1882), but including index for Volumes Nos. 1 to 35, and Nos. 36 to 40,	75
VI. Nine volumes, bound in half-morocco, from No. 1 (1873) to No. 9 (1881),	25

Applications should be addressed to Joseph Struthers, Secretary, 29 West 39th Street, New York, N. Y.

The Emmons Research Fellowship of Economic Geology.

The Committee named below has been formed by friends of Samuel Franklin Emmons, late of the United States Geological Survey, to consider the best method of perpetuating his name. It has been decided that the memorial to him shall take the shape of a Research Fellowship, to be known as the Samuel Franklin Emmons Research Fellowship of Economic Geology, which is to be administered by Prof. James F. Kemp, of Columbia University, New York. Subscriptions are invited by his friends to this fund, which the Committee has fixed at $25,000.

Members of the Institute who desire to contribute to this fund will please communicate with the Treasurer, Benjamin B. Lawrence, 60 Wall Street, New York.

The Committee consists of the following:

GEORGE OTIS SMITH, Director, U. S. Geological Survey, Washington, D. C.
H. L. SMYTH, Harvard University, Cambridge, Mass.
JAMES DOUGLAS, 99 John Street, New York, N. Y.
J. A. HOLMES, Director, Bureau of Mines, Washington, D. C.
JAMES F. KEMP, Columbia University, New York, N. Y.
F. W. BRADLEY, San Francisco, Cal.
J. PARKE CHANNING, 42 Broadway, New York, N. Y.
SEELEY W. MUDD, 1001 Central Building, Los Angeles, Cal.
D. W. BRUNTON, Denver, Colo.
H. FOSTER BAIN, 420 Market Street, San Francisco, Cal.
T. A. RICKARD, London, England.
B. B. LAWRENCE, 60 Wall Street, New York, N. Y.

How to Use the "Transactions" of the Institute.

Buy a copy of the Complete Analytical and Alphabetical Index of Volumes I. to XXXV., inclusive; also the new Index of Volumes XXXVI. to XL.

Whether you do or do not own a full set of the *Transactions*, these Indexes will make all of the material contained in the forty volumes available at once without detailed research into each volume separately. Moreover, an easy search will show what particular papers you need to know more about, and perhaps to study. Thus, any person possessing these Indexes can ascertain at once what has been published in the *Transactions* on a given question, and can learn, by writing to the Secretary, what is its nature, whether it is still to be had in pamphlet form, where it can be consulted in a public library, at what cost it can be copied by hand, etc., etc.

In short, to those who own complete sets of the *Transactions*, these Indexes will be a great convenience; but to those who do not, they will be a professional necessity.

The Index Volumes I. to XXXV. is an octavo of 706 pages, containing more than 60,000 entries, duly classified with sub-headings, and including abundant cross-references. The limited edition is becoming exhausted. The new Index, Volumes XXXVI. to XL., supplementing the Index Volumes I. to XXXV., brings the classified references up to the date of Volume XL., June, 1910. Prices: Index Volumes I. to XXXV., bound in cloth, $5; bound in half-morocco, to match the *Transactions*, $6. Index Volumes XXXVI. to XL., bound in cloth, $1.50; bound in half-morocco, $2.50. The delivery charges will be paid by the Institute on receipt of the above price.

Hydrographic Chart.

Owing to the great value to hydrographers of the chart contained in the paper, A Graphic Solution of Kutter's Formula, by L. I. Hewes and Joseph W. Roe (*Bulletin No.* 29, May, 1909, p. 454), a special edition for office or field use has been printed on durable cloth. Copies of this separate chart may be obtained, at a cost of 50 cents each, on application to the office of the Secretary of the Institute.

LIBRARY.

AMERICAN INSTITUTE OF ELECTRICAL ENGINEERS.
AMERICAN SOCIETY OF MECHANICAL ENGINEERS.
AMERICAN INSTITUTE OF MINING ENGINEERS.
UNITED ENGINEERING SOCIETY.

WILLIAM P. CUTTER, Librarian.

The libraries of the above-named Societies are open from 9 A.M. to 9 P.M. on all week-days, except holidays, from September 1 to June 30, and from 9 A.M. to 6 P.M. during July and August.

The Library contains about 50,000 volumes, including sets of technical periodicals and the publications of scientific and technical societies.

The members of the Institute, with few exceptions, are by the very nature of their profession forced to spend a large portion of their time in localities isolated from sources of information. To such members the Library can render valuable service through correspondence, and letters requesting information will receive special attention. The Library is prepared to furnish references and copies of articles on mining and metallurgical subjects; to determine, if possible, the existence of mining-maps, and to furnish general information as to the geology and mineral resources of all countries as far as these resources are known and published.

It is hoped that the members of the Institute will avail themselves freely of this special service. The Library will welcome inquiries on engineering subjects, and furnish information as far as such information is to be obtained.

All communications should be made as definite as possible so that the information received may be what is desired and not include collateral matter which may not be of interest. In this way the time spent in searching for such collateral matter will be saved, and as a result the information will be sent more promptly and in more usable shape.

The members of the Institute can be of service to the Library by forwarding copies of mining-reports, maps privately issued, and similar material, which will be classified, indexed, and made available to other members.

Suggestions for additions to the Library, either by purchase or personal solicitation as gifts, will be welcomed. It is hoped that members while in the city will use the Library freely, and assurance is given that most careful service will be rendered to them.

Library Accessions.

Aug. 1 to Aug. 31, 1912.

[Copies of the list of additions to the Libraries of the American Society of Mechanical Engineers and the American Institute of Electrical Engineers can be obtained on application to the Secretary of the American Institute of Mining Engineers.]

ALBUM OF DRAWINGS RELATING TO BLAST-FURNACE CONSTRUCTION. (In Russian). By M. A. Pavloff. Ekaterinoslav, 1902. (Purchase.)

ALBUM OF DRAWINGS RELATING TO THE MANUFACTURE OF OPEN-HEARTH STEEL. Ed. 2. By M. A. Pavloff. St. Petersburg, 1908. (Purchase.)

AMERICAN CHEMICAL SOCIETY. Directory, 1912. N. p., 1912. (Exchange.)

AMERICAN ELECTROCHEMICAL SOCIETY. Transactions. Vol. XXI., 1912. South Bethlehem, 1912. (Exchange.)

AMERICAN FOUNDRYMEN'S ASSOCIATION. Transactions. Vols. XIX., XX. Watchung, 1911, 1912. (Exchange.)

AMERICAN INSTITUTE OF CHEMICAL ENGINEERS. Transactions. Vol. IV., 1911. New York, 1912. (Exchange.)

ARCHITECTS' BENEVOLENT SOCIETY. Report of the Council, 1911. List of Donors and Annual Subscribers by-laws. London, 1912. (Gift of Royal Institute of British Architects.)

BERGWERKS INSPEKTION IN OSTERREICH, 1909. Wien, 1912. (Exchange.)

BIBLIOGRAPHIE DER FREMDSPRACHIGEN ZEITSCHRIFTENLITERATUR. Band II., 1911. By F. Dietrich. Leipzig, 1912. (Purchase.)

BOSTON SOCIETY OF CIVIL ENGINEERS. Year Book, 1912–13. Boston, 1912. (Exchange.)

BRITISH ASSOCIATION FOR THE ADVANCEMENT OF SCIENCE. Report, 1910, 1911. London, 1911–1912. (Purchase.)

BRITISH COLUMBIA. MINISTER OF MINES. Annual Report, 1911. Victoria, 1912. (Exchange.)

CANADA. MINES DPARTMENT. Catalogue of Publication of the Mines Branch, 1907–1911. Ottawa, 1912. (Exchange.)

——— Memoir, Nos. 24–E., 28. Ottawa, 1912. (Exchange.)

CARLYLE OIL FIELD AND SURROUNDING TERRITORY. By E. W. Shaw. (Extract from Bulletin 20, Illinois State Geological Survey.) Urbana, 1912. (Exchange.)

CARNEGIE MUSEUM. Annual Report of the Director, 15th, 1912. Pittsburgh, 1912. (Gift of Carnegie Museum.)

——— Founder's Day, 1912. Pittsburgh, 1912. (Gift of Carnegie Museum.)

COAL RESOURCES OF NEW SOUTH WALES. By E. F. Pittman. Sydney, 1912. (Exchange.)

COURS DE METALLURGIE DES MÉTAUX AUTRES QUE LE FER. Par Eug. Prost. Paris, 1912. (Purchase.)

EFFECT OF STEMMING ON THE EFFICIENCY OF EXPLOSIVES. (Technical Paper No. 17, U. S. Bureau of Mines.) Washington, 1912. (Exchange.)

ERFORSCHUNG UND ABWENDUNG DER KOHLENSTAUB-EXPLOSIONEN. Zweiter Beitrag. By A. Padour. Teplitz-Schönau, 1911. (Purchase.)

FABRICATION DE L'ACIER. Par H. Noble. Paris, 1905. (Purchase.)

DIE FLUSSIGEN HEIZMATERIALIEN UND IHRE ANWENDUNG. By F. A. Rossmassler. Wien-Leipzig, 1910. (Purchase.)

FRANKLIN INSTITUTE SCHOOL OF MECHANIC ARTS. Announcement, 89th year, 1912–13, Evening Courses and Classes. Philadelphia, 1912. (Gift of Franklin Institute.)

GEOLOGY AND ORE-DEPOSITS OF THE BUTTE DISTRICT, MONTANA. By W. H. Weed. (Professional Paper No. 74, U. S. Geological Survey.) Washington, 1912. (Exchange.)

GEOLOGY OF STEEPROCK LAKE, ONTARIO. (Memoir No. 28, Canada Mines Department.) Ottawa, 1912. (Exchange.)

GEWERBLICHE VERGIFTUNGEN DEREN VORKOMMEN, ERSCHEINUNGEN, BEHANDLUNG VERHÜTUNG. By J. Rambousek. Leipzig, 1911. (Purchase.)

GREAT BRITAIN PATENT OFFICE LIBRARY. Subject List of Works on Mineral Industries. Pts. 2–3. London, 1912. (Exchange.)

HANDBUCH DER MINERALCHEMIE. Pt. 4. By C. Doelter. Dresden, 1912. (Purchase.)

IGNITION OF GAS BY MINIATURE ELECTRIC LAMPS WITH TUNGSTEN FILAMENTS. (Technical Paper No. 23, U. S. Bureau of Mines.) Washington, 1912. (Exchange.)

L'INDUSTRIE HOUILLERE DU DEPARTMENT DU NORD. By M. Mettrier. N. p., n. d. (Purchase.)

INTRODUCTION TO THE STUDY OF FUEL. By F. J. Brislee. London, 1912. (Purchase.)

IRIDOSMINE FROM THE NEW RIETFONTEIN MINES: ITS OCCURRENCE, ANALYSIS, AND GENESIS. By C. B. Horwood. (From the Transactions of the Geological Society of South Africa, Vol. XV, 1912.) (Gift of Author.)

IRON AND STEEL INSTITUTE. Carnegie Scholarship Memoirs. Vol. IV. London, 1912. (Exchange.)

——— Charter, By-Laws, and List of Members, July, 1912. London, 1912. (Exchange.)

JAPAN. Financial and Economic Annual, 12th, 1912. Tokyo, 1912. (Gift of Japan Department of Finance.)

KEROSENE SHALE DEPOSITS OF NEW SOUTH WALES. (Memoirs, Geology, No. 3. New South Wales Geological Survey.) Sydney, 1903. (Exchange.)

DER "MABUKI"-PROZESS. By Voichi Okada. Freiberg in Sachsen, 1911. (Purchase.)

MANCHESTER ASSOCIATION OF ENGINEERS. Transactions, 1910–1911. Manchester, 1911. (Exchange.)

MAGAZINES AND THAW HOUSES FOR EXPLOSIVES. (Technical Paper No. 18, U. S. Bureau of Mines.) Washington, 1912. (Exchange.)

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METALLURGICAL MANUAL OF IRON AND STEEL. By Horace Allen. London, 1911. (Purchase.)

METALLURGISCHE UND TECHNOLOGISCHE STUDIEN AUF DEM GEBIETE DER LEGIERUNGS-INDUSTRIE INSBESONDERE ÜBER DAS AUSGLÜHEN VON METALLEN UND LEGIERUNGEN. By Max Weidig. Berlin, 1912. (Purchase.)

MICHIGAN. DEPARTMENT OF LABOR. Annual Report, 29th. Lansing, 1912. (Exchange.)

MINING LAW OF THE BRITISH EMPIRE. By C. J. Alford. London, 1906. (Purchase.)

MINING SOCIETY OF NOVA SCOTIA. Journal. Vol. XVI. Halifax, 1911. (Exchange.)

MITTEILUNGEN AUS DEM EISENHÜTTENMANNISCHEN INSTITUT DER KONIGL. TECHN. HOCHSCHULE AACHEN. By F. Wüst. Vols. I–IV. Halle a/S., 1906, 1908, 1909, 1911. (Purchase.)

NEW ORLEANS. SEWERAGE AND WATER BOARD. 24th Semi-Annual Report. New Orleans, 1911. (Gift of Sewerage and Water Board.)

NEW SOUTH WALES. GEOLOGICAL SURVEY. Memoirs. Geology, No. 3. Sydney, 1903. (Exchange.)

NEW ZEALAND INSTITUTE. Transactions and Proceedings. Vol. XLIV., 1911. Wellington, 1912. (Exchange.)

ONONDAGA FAUNA OF THE ALLEGHENY REGION. By E. M. Kindle. (Bulletin No. 508, U. S. Geological Survey.) Washington, 1912. (Exchange.)

DER PRAKTISCHE HOCHOFENBETRIEB. By Theodor Bender. Halle a/S., 1910. (Purchase.)

PRELIMINARY REPORT ON THE CLAY AND SHALE DEPOSITS OF THE WESTERN PROVINCES. By H. Ries and J. Keele. (Memoir No. 24-E, Canada Mines Department.) Ottawa, 1912. (Exchange.)

RESULTS OF SPIRIT LEVELING IN ALABAMA, 1911. (Bulletin No. 517, U. S. Geological Survey.) Washington, 1912. (Exchange.)

ROYAL SOCIETY OF VICTORIA. Proceedings. Vol. XXIV., Pt. II. Melbourne, 1912. (Exchange.)

SOCIETY FOR THE PROMOTION OF ENGINEERING EDUCATION. List of Members, 1912. Lancaster, 1912. (Gift of R. W. Raymond.)

STIELER'S HAND ATLAS. Gotha, 1912. (Purchase.)

STORY OF COAL AND IRON IN ALABAMA. By Ethel Armes. Birmingham, 1910. (Purchase.)

SURVEYORS' INSTITUTION. List of Members, 1912. Westminster, 1912. (Exchange.)

DIE TECHNIK IN DER EISENGIESSEREI UND PRAKTISCHE WISSENSCHAFT. Ed. IV. By A. Messerschmitt. Essen a. d. Ruhr, 1909. (Purchase.)

THEORIE UND PRAXIS DER STAUBVERDICHTUNG UND DER REINIGUNG UND ENTSTAUBUNG VON GASEN. By C. Guillemain. Halle a/S., 1911. (Purchase.)

TRANSMISSION OF HEAT INTO STEAM BOILERS. (Bulletin No. 18, U. S. Bureau of Mines.) Washington, 1912. (Exchange.)

TREATISE ON CERAMIC INDUSTRIES. By Emile Bourry. London, 1911. (Purchase.)

TREATISE ON HYDRAULICS. Ed. 9. By Mansfield Merriman. New York, J. Wiley & Sons, 1912. Price $4 net. (Gift of Publishers.)

[NOTE.—Since the first publication of this treatise, in 1889, it has gone through eight editions, and in 1903–1910, the eighth edition was printed eight times, with annual changes and corrections. The total issue of 37,000 copies—an extraordinary number for a work of this character—is a striking evidence of the value of the book, as recognized by those for whose use it was intended. Probably no other one of the long list of treatises which have come from the pen of this distinguished veteran in technical authorship has received so wide a welcome. One obvious reason is the circumstance that, as compared with his works on other engineering subjects (even including the American Civil Engineers' Pocket-Book, of which he has been since 1908 editor-in-chief), this one worthily occupies a field not crowded with competitors.

But, in the progress of engineering, there must come a time for every such treatise when successive additions and modifications will no longer keep it up-to-date. Recognizing this inevitable necessity, Prof. Merriman has reconstructed the whole book, with the competent aid of his son Mr. Thaddeus Merriman (himself also a member of the American Society of Civil Engineers), so that this ninth edition is practically a new treatise, preserving what is still valuable in older theory and practice, while giving adequate space and effect to the modern developments of both. It is needless to say that the work has been well done. Among the new topics treated for the first time are hydraulic instruments and observations (including methods of measuring pressures and velocities) and pumps and pumping (including a discussion, from the standpoint of hydraulics, of the various machines for raising water). But there are many other new subjects, not treated, like these, in separate chapters, but dispersed through the text, illustrated with problems, and accompanied with an increased abundance of references to technical literature.—R. W. R.]

U. S. AGRICULTURE DEPARTMENT. Year Book, 1911. Washington, 1912. (Exchange.)

U. S. GEOLOGICAL SURVEY. Bulletin Nos. 508, 517. Washington, 1912. (Exchange.)

——— Professional Paper No. 74. Washington, 1912. (Exchange.)

U. S. MINES BUREAU. Bulletin No. 18. Washington, 1912.

——— Technical Paper Nos. 17, 18, 23. Washington, 1912. (Exchange.)

U. S. ORDNANCE DEPARTMENT. Report of the Tests of Metals and other Materials, 1911. Washington, 1912. (Exchange.)

VICTORIA. GEOLOGICAL SURVEY. Bulletin Nos. 18–23. Melbourne, 1906–07, (Exchange.)

——— MINES DEPARTMENT. Memoirs. Nos. 4–10. Melbourne, 1907–11. (Exchange.)

HONORABLE PETER WHITE. A Biographical Sketch of the Lake Superior Iron Country. By R. D. Williams. Cleveland, 1907. (Purchase.)

WHO'S WHO IN OHIO STATE UNIVERSITY ASSOCIATION, 1912. Columbus, 1912. (Exchange.)

ZEITSCHRIFT FÜR ANORGANISCHE CHEMIE. Vols. 70–76, 77, Pt. 1. Leipzig, 1911–12. (Purchase.)

ZUSAMMENSETZUNG DER GEBRÄUCHLICHEN METALLEGIERUNGEN. By Wilhelm Kaiser. Halle a/S., 1911. (Purchase.)

GIFT OF TECHNISCHE HOOGESCHOOL A DELFT.

SIEGER, W. Het Aethyleeren van Chloorbenzol. Amsterdam, 1912.

VAN AMSTEL, J. E. De Temperatuursinvloed op Physiologische Processen der Alcoholgist. Amsterdam, 1912.

VERHOECKX, P. M. Proeve eener Theorie van het Roteerend Magnetisch Veld. 's Gravenhage, 1912.

WENTHOLT, L. R. Stranden en Strandverdediging. N. p., n. d.

——— ——— Atlas. N. p., n. d.

TRADE CATALOGUES.

INGERSOLL-RAND CO., New York, N. Y. *Golden Opportunities*, Jan., 1912. Collected letters of recommendation on the Leyner drill-sharpener. 48 pages.

INTERNATIONAL OXYGEN CO., New York, N. Y. I. O. C. system for generating oxygen and hydrogen. 16 pages.

MESTA MACHINE CO., Pittsburg, Pa. Plant and products of Mesta Machine Co. 47 pages.

United Engineering Society Library.

OHIO RAILROAD COMMISSION. Report, 1911. Springfield, 1911. (Gift of Ohio Railroad Commission.)

ST. LOUIS PUBLIC LIBRARY. Annual Report, May 1, 1911–April 30, 1912. St. Louis, 1912. (Gift of St. Louis Public Library.)

"TITANIC" DISASTER. Hearings before a subcommittee of the Committee on Commerce, United States Senate, 62d Congress, Second Session. Washington, 1912. (Gift of U. S. Senate.)

GIFT OF ASSOCIATION OF AMERICAN STEEL MANUFACTURERS.

STANDARD SPECIFICATIONS FOR CONCRETE REINFORCEMENT BARS ROLLED FROM BILLETS, as adopted 1910. Revised 1912.

STANDARD SPECIFICATIONS FOR STRUCTURAL AND BOILER STEEL, as adopted Aug. 9, 1895. Revised 1896, 1902, 1903 and 1912.

STANDARD SPECIFICATIONS FOR RAIL STEEL CONCRETE REINFORCEMENT BARS, as adopted 1912.

MEMBERSHIP.

NEW MEMBERS.

The following list comprises the names of those persons elected as members who accepted election during the month of August, 1912:

Members.

DALTON, MICHAEL P., Mill Supt..Mullan, Idaho.
GWINN, JAMES W., Min. Engr.. Kellogg, Idaho.
HOLDERER, GEORGE B., Min. Engr..................165 Broadway, New York, N. Y.
INOUYÉ, TADASHIRO, Prof. of Mining, Tokyo Imperial University,
83 Shimo osaki-machi, Yebara-gori, Tokyo fu, Japan.
MILLARD, JOHN W., Prospector, Société Internationale Minière et Forestière du Congo, Mission Minière Bas Congo, Boma, Congo Belge, W. Africa.
MORGAN, CHARLES G., Cons. Engr..Roslyn, Wash.
PEATTIE, CLARENCE G., Min. Engr........Empire Steel & Iron Co., Oxford, N. J.
SAUNDERS, T. SKEWES, Min. Engr., Teziutlan Copper Co., Aire Libre, Pue., Mexico.
SCHIFFNER, OTTO E., Min. Engr..................................Dale, via Amboy, Cal.
SEIBERT, PERCY A., Mgr., Andes Tin Co., La Paz, Bolivia, So. America, via Panama.
VAN WINKLE, CHARLES T., Min. Engr.........Scott Bldg., Salt Lake City, Utah.
WILSON, ARDEN MARTIN, Mine Mgr.................P. O. Box 2[illegible]2, Telluride, Colo.

Associate.

MARTIN, SAMUEL G., Mining..........................321 Boston Bldg., Denver, Colo.

CANDIDATES FOR MEMBERSHIP.

The following persons have been proposed during the month of August, 1912, for election as members of the Institute. Their names are published for the information of members and associates, from whom the Committee on Membership earnestly invites confidential communications, favorable or unfavorable, concerning these candidates. A sufficient period (varying in the discretion of the Committee, according to the residence of the candidate) will be allowed for the reception of such communications, before any action upon these names by the Committee. After the lapse of this period, the Committee will recommend action by the Council, which has the power of final election.

Members.

Herbert Watson Alden, Detroit, Mich.
Proposed by H. M. Howe, Henry D. Hibbard, Bradley Stoughton.

Robert Alexander Bull, Granite City, Ill.
Proposed by Bradley Stoughton, Richard Moldenke, Charles F. Rand.

James Pascoe Caddy, Kalgoorlie, West Australia.
Proposed by H. E. Vail, John A. Agnew, J. W. Sutherland.

F. C. Frey, Java, Dutch East Indies.
Proposed by Pope Yeatman, H. H. Webb, R. M. Catlin.

Carl Goedicke, San Antonio, Texas.
Proposed by D. W. Reckhart, W. N. Small, J. M. Wines.

G. F. Hendricks, Porcupine, Ont., Can.
 Proposed by Louis D. Huntoon, Bradley Stoughton, Arthur H. Elliott.
Masaaki Hioki, Akita, Japan.
 Proposed by Edward L. Young, S. Yamanouchi, T. Kawamura.
John H. Jones, Spokane, Wash.
 Proposed by L. K. Armstrong, J. V. Richards, Francis A. Thomson.
Ira Beaman Joralemon, Warren, Arizona.
 Proposed by Max Roesler, Gerald Sherman, A. R. Ledoux.
William Howard Lamb, Jr., Corona, Ala.
 Proposed by C. A. Moffett, John R. Pill, Milton H. Fies.
E. C. Limbach, Loomis, Wash.
 Proposed by L. K. Armstrong, J. V. Richards, J. C. Haas.
Arthur G. McKee, Cleveland, Ohio.
 Proposed by S. W. Croxton, M. McMurray, D. T. Croxton.
Lee Clayton Morganroth, Pittsburg, Pa.
 Proposed by Charles F. Rand, Bradley Stoughton, Jennings S. Cox, Jr.
Arthur Notman, Bisbee, Ariz.
 Proposed by Max Roesler, Roger T. Pelton, Walter Douglas.
Vernon Stone Rood, Bingham Canyon, Utah.
 Proposed by Stanley C. Sears, J. F. Kemp, E. P. Jennings.
Bert Norris Sharp, Republic, Wash.
 Proposed by S. H. Richardson, J. V. Richards, L. K. Armstrong.
Jasper Holmes Sheadle, Cleveland, Ohio.
 Proposed by M. McMurray, S. W. Croxton, D. T. Croxton.
Francis J. Simington, Chicago, Ill.
 Proposed by G. H. Cox, Arthur J. Hoskin, S. W. Berger.
Alfrid William Skerritt, Doten, Korea.
 Proposed by A. H. Collbran, J. F. Manning, James J. Martin.
James W. Tremlett, Bisbee, Ariz.
 Proposed by Walter Douglas, Gerald Sherman, Roger T. Pelton.
Milton F. Webster, Spokane, Wash.
 Proposed by L. K. Armstrong, J. V. Richards, J. C. Haas.
Frederick Charles Wilson, Felton, Cuba.
 Proposed by Charles F. Rand, Jennings S. Cox, Jr., James E. Little.
Howard Malcolm Yeager, Wallsend, Ky.
 Proposed by Eli T. Conner, H. McKean Conner, G. M. Shoemaker.

Associates.

Benno Elkan, New York, N. Y.
 Proposed by Benjamin B. Lawrence. Thomas W. Stiles, J. E. McAllister.
Nelson Sutro Greensfelder, Chewelah, Wash.
 Proposed by J. V. Richards, L. K. Armstrong, J. C. Haas.

CHANGES OF ADDRESS OF MEMBERS.

The following changes of address of members have been received at the Secretary's office during the month of August, 1912. This list, together with the lists published in *Bulletin* Nos. 63 to 68, March to August, 1912, and the foregoing list of new members, therefore, supplements the annual list of members corrected to Feb. 1, 1912, and brings it up to the date of Sept. 1, 1912.

ALLEN, ROY H.....Care Sierra Plata Mining Co., Villa Escobedo, Chih., Mexico.
BERGER, W. F. B...825 Symes Bldg., Denver, Colo.
BOND, JOSIAH, Mining..Tolnachi Mine, Alto, Ariz.
BOTSFORD, ROBERT S., Nicolaevo Povdinsk Estate, Kyshtim, Vyia Station,
 Bogoslovsk, R. T., Govt. of Perm, Russia.
BOWMAN, FRANCIS C..Trojan, S. Da.
BRENDEL, ROBERT.....................Kaiser Wilhelm Str. 24, Hannover, Germany.

Burrall, Frederick P..........509 W. 121st St., New York, N. Y.
Clark, Fred..........410 Hearst Bldg., San Francisco, Cal.
Copeland, Fredk. K..........413 Peoples Gas Bldg., Chicago, Ill.
Crowley, T. Irwin..........Dome Mine, So. Porcupine, Ont., Canada.
Dake, Walter M., Jr., Pittsburgh-Silver Peak G. M. Co.,
P. O. Box 63, Blair, Nev.
Dixon, A. Faison.....101 Southampton Row, Russell Sq., London, W., England.
Drake, Noah F..........Fayetteville, Ark.
Edwards, J. Warner..........318 14th St., Denver, Colo.
Ellam, Albert S..........30 Birkbeck Road, Acton, London, W., England.
Gardner, E. D., Min. Engr..........Forest Service, Missoula, Mont.
Gillette, Halbert P..........608 S. Dearborn St., Chicago, Ill.
Green, George G..........Battelle, De Kalb Co., Ala.
Hamfeldt, George K..........S. Hamngatan 17, Gothenbourg, Sweden.
Harris, Harold J..........Thoburn P. O., near Victoria, B. C., Canada.
Heberlein, Kuno B., Genl. Mgr., Cia. Minera de Penoles, S. A.,
Apartado 733, Mexico City, Mexico.
Hedges, Joseph H..........Apartado 25, Guanajuato, Mexico.
Henderson, J. McC..........Springfield, Lane Co., Ore.
Hicks, Charles M..........Wytheville, Va.
Holberton, Walter T., Mgr., Société Française de Mines de Lenire,
Collahuasi, La Grande, Antofagasta, Chile, S. A.
Huang, Saosan Ken, Min. Engr., Cambria Steel Co., P. O. Box 202, Johnstown, Pa.
Humphrey, John M..........458 S. Franklin St., Wilkes-Barre, Pa.
Imhoff, Wallace G..........P. O. Box 515, Iron River, Mich.
Janney, T. Gordon..........Leesburg, Va.
Kaeding, Henry B., Care Am. Inst. of Mining Engineers,
29 W. 39th St., New York, N. Y.
Kennedy, John S..........Chambersburg, Pa.
Kenney, Henry S..........Box 3541, Johannesburg, Transvaal, So. Africa.
Kirchen, Charles..........Manhattan, Nev.
Kirk, Morris P..........Newhouse Bldg., Salt Lake City, Utah.
Leggett, Thomas H., Cons. Engr., Min. Dept., Am. Smelt. and Refin. Co.,
165 Broadway, New York, N. Y.
Lewis, Robert S., Asso. Prof. of Mining, University of Utah,
Salt Lake City, Utah.
Lihme, Christian B..........Maraine Hotel, Highland Park, Ill.
Macfarlane, Rienzi W., Min. Engr......Apartado 102, Parral, Chih., Mexico.
Marshall, Emory M..........McConnell Mine, Mason, Nev.
Merrill, Monroe E..........6911 Franklin Ave., Hollywood, Cal.
Mills, Kenneth..........1305 16th St., N. W., Washington, D. C.
Morton, Erle D..........6001 Eleanor Ave., Los Angeles, Cal.
Porter, Jesse C..........Abangarez Gold Fields, Abangarez, Costa Rica, C. A.
Ralston, W. C..........905 Royal Insurance Bldg., San Francisco, Cal.
Rehfuss, Louis A..........Reno, Nev.
Rickard, Arthur J..........Stokeamber House, Penryn, Cornwall, England.
Robbins, Hallet R..........P. O. Box 51, Seattle, Wash.
Scheffler, F. G., Cie. Mines de Compinas, Encruzilhada,
Rio Grande do Sul, Brazil, So. Amer.
Schrader, Erich J., Genl. Mgr., Pittsburgh-Dolores M. Co.,
Pine Grove, via Yerington, Nev.
Scott, W. M. Henderson..........Instructed to hold all mail.
Sears, Stanley C., Min. Engr. and Met., Genl. Mgr.,
Utah Apex Mining Co., Bingham Canyon, Utah.
Sheafe, Harry J..........El Dorado, Cal.
Sweetser, Arthur L., New York & Honduras Mining Co.,
Rosario, Honduras, C. A.
Thorne, William E..........408 Salisbury House, London, E. C., England.
Tombo, Carl..........General Delivery, Marquette, Mich.
Turner, Henry W..........708 Mills Bldg., San Francisco, Cal.
Tuttle, Arthur L..........Care City Brewery, San Antonio, Texas.
Ward, William F..........416 Ernest & Cranmer Bldg., Denver, Colo.
Wash, Edwin R., Chief Engr..........Mascot Copper Co., Des Cabezas, Ariz.
Williams, Ralph B.....American Smelting & Refining Co.,
Angangueo, Mich., Mexico.

ADDRESSES OF MEMBERS AND ASSOCIATES WANTED.

Name.	Last Address of Record, from which Mail has been Returned.
Bates, Mowry,	Hughes Porcupine Mines, Ltd., Porcupine, Ont., Canada.
Baxter, Francis K., Jr.,	423 Wells Fargo Bldg., San Francisco, Cal.
Cook, Edward H.,	Minas Birimoa, S. A., Birimoa, via Canelas, Dur., Mexico.
Dane, Edward G.,	43 Arundel Mansions, London, England.
Danforth, A. H.,	Cotopaxi, Colo.
David, W. M.,	Care C. G. Hussey & Co., Second Ave., Pittsburg, Pa.
Dyott, George M.,	50 Church St., New York, N. Y.
Fergusson, Hugh B.,	216 Loo Bldg., Vancouver, B. C., Canada.
Fitzgerald, Thomas F. M.,	211 Sharon Bldg., Salt Lake City, Utah.
Furness, James W.,	Coffee, Trinity Co., Cal.
Gmelin, Ernest,	Apartado 57, Aguascalientes, Mexico.
Haggen, Edward A.,	220 Pemberton Blk., Victoria, B. C., Canada.
Higgins, Edwin,	Randsburg, Cal.
Hilson, Cleaveland,	Livingston, Mont.
Hollis, R. W.,	Silverton, Colo.
Johnson, Dion L.,	325 Water St., Pittsburg, Pa.
Lampshire, John O.,	Vulture Mine, Wickenburg, Ariz.
Leavell, John H.,	Buffalo Mine, Cobalt, Ont., Canada.
Le Noir, Frank H.,	Box 16, Mt. Bullion, Cal.
McDougall, Wallace D.,	20 Bedford Place, Russell Sq., London, Eng.
Mitinskij, Alexander N.,	V. O. 12, line 33, St. Petersburg, Russia.
Moore, Roy W.,	P. O. Box 48, Velasco, Tex.
Munroe, Martin,	Bengal Coal Co., Murulidih, Mohada, B. N. Ry., Bengal, India.
Nelson, D. W. C.,	Baker City, Ore.
Nobs, Frederick W.,	Negociacion Minera Santa Maria de Guadalupe y Anexas, S. A., Minillas, Zac., Mexico.
Oldfield, Frank W.,	Mexican Mines Co., Bolaños, Jal., Mexico.
Pearson, William R.,	628 W. 114th St., New York, N. Y.
Perks, Harry B.,	419 Board of Trade Bldg., Portland, Ore.
Peterson, Frank,	H. W. Hellman Bldg., Los Angeles, Cal.
Rathborne, Merwyn R. W.,	Amargosa, via Las Vegas, Nev.
Rhew, James W.,	Cia. Minera y Exploradora de Ventanas, S. A., Ventanas, Dur., Mex.
Sheldon, Waldo,	Urique, Chih., Mexico.
Short, Frank R.,	Carson City, Nev.
Thornton, Edward T.,	Apartado 30, Matehuala, S. L. P., Mexico.
Twynam, Henry,	O. K. Copper Mine, Cairns, No. Queensland, Australia.
Vinicombe, Robert E. B.,	Post Restante, Vladikafkas, So. Russia.
Watson, Ralph W.,	Calloo, Utah, Clifton Mail box.
Webster, Erastus H.,	Hotel Cosmopolita, Guadalajara, Jal., Mexico.
White, W. P.	10 Bernard St., Spokane, Wash.

NECROLOGY.

The deaths of the following members were reported to the Secretary's office during the month of August, 1912:

Date of Election.	Name.	Date of Decease.
1900.	*Chisholm, John,	May 12, 1912.
1894.	*Jackson, T. Moore,	February 3, 1912.
1904.	*McLaughlin, Harold M.,	July 30, 1912.
1902.	*Poindexter, Charles L.,	July 29, 1912.

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* Member.

SUBJECT TO REVISION.

[TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS.]

The Decomposition of Metallic Sulphates at Elevated Temperatures in a Current of Dry Air.

BY H. O. HOFMAN AND W. WANJUKOW, MASS. INSTITUTE OF TECHNOLOGY, BOSTON, MASS.

(New York Meeting, February, 1912.)

I. INTRODUCTION.

IN the metallurgical treatment of most metallic sulphides it is usually necessary to carry on a roasting-operation. In some cases the raw ore will have to be roasted; in others the intermediary product, matte, produced in smelting the ore; again, both ore and matte may have to be subjected to oxidation at elevated temperatures.

Most metallic sulphides, when given an oxidizing-roast, pass through the state of sulphate before they are converted into oxides. All metallic sulphates are completely decomposed by heat, with the exception of those of lead and bismuth, but the temperatures at which decompositions take place vary with the metals. Some metallic sulphates are converted directly into oxides, others first form basic salts. Alkali-earth sulphates show a behavior similar to that of metallic sulphates, while alkali sulphates do not part with their sulphur trioxide even at elevated roasting-temperatures.

The older data on the decomposition of metallic sulphates by heat are only approximations. Thus Kerl[1] gives the order in which metallic sulphates are decomposed with increasing temperatures as $(Ag_2, Fe, Cu, Zn, Ni, CO, Mn, Pb)\ SO_4$. More recent statements give definite temperatures and modify the formerly accepted order of decomposition. But the temperatures given by various observers do not agree. The reasons for this are to be found mainly in the method and manner of carrying out experiments.

Four methods have been used: 1. Heating in an evacuated tube; 2, heating in a tube closed at one end, without air-current; 3, heating in a crucible with air-current; and 4, heating in an open tube with air-current.

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[1] *Grundriss der Metallhüttenkunde*, p. 70 (Leipzig, 1881).

1. *Heating in an Evacuated Tube.*—This method is the one followed by Wöhler-Plüddemann-Wöhler.[2] A tube charged with the sulphate to be examined is evacuated, heated to different temperatures, and held at each temperature until equilibrium has been reached in the reversible equation $MSO_4 \rightleftarrows MO + SO_3$. The pressure exerted by the SO_3, resp. $SO_2 + O$, set free at each temperature and read in millimeters of quicksilver on a gauge, furnishes the means for ascertaining the degree of decomposition of the sulphate. The decomposition at atmospheric pressure is the one at which the pressure of the gas in the tube has reached 760 mm. In order to determine whether the sulphate by heating is converted direct into oxide or first forms one or more basic salts, it is necessary to heat the normal sulphate first to a given temperature in a current of air to expel some of the SO_3, and then in vacuo for determining the dissociation-pressures at the same temperature to which the normal sulphate had been brought. If the pressure-readings of both experiments give the same figures, no basic salt has been formed up to the temperature to which the normal salt had been heated in a current of air. In this manner, samples of normal salt will have to be heated in air to different temperatures, and the partial decompositions finished in vacuo. A comparison of the dissociation-curve of the normal salt with those of the partly decomposed salts will show at a glance whether basic salts are formed or not. We investigated the following metallic sulphates: $Fe_2O_3.3SO_3$; $Al_2O_3.3SO_3$; $Cr_2O_3.3SO_3$; $CuO.SO_3$; $CeO_2.3SO_3$; $ThO_2.2SO_3$; $TiO_2.SO_3$; $ZnO.SO_3$. The data for the sulphates of metallurgical interest have been plotted in Fig. 1, in which the ordinate gives the degrees centigrade scale, and the abscissa the pressures of SO_3 in millimeters of quicksilver. In order to translate the pressures of SO_3 into per cent. by volume of SO_3, and thus render the data comparable with roasting-gases, a second scale has been drawn on the abscissa, in which 760 mm. pressure has been made equal to 100 divisions, presupposing that SO_3 is not split into SO_2 and O; thus 1 volume of SO_3

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[2] *Berichte der deutschen chemischen Gesellschaft*, vol. xli., pt. 1, No. 4, p. 703 (1908).

will equal 7.6 mm. pressure SO_3. These data, while of much scientific interest, cannot be well applied to the decomposition of metallic sulphates in a roasting-furnace, as in the latter the gases are drawn off continuously, and the partial pressure of SO_3 is so small as to have practically no influence whatever upon the decomposition.

2. *Heating in a Tube Closed at One End without a Current of Air.*—This method has been recently used by Friedrich and Blickle.[3] The apparatus consist of a vertical porcelain test-tube, 3.937 in. long, 0.7078 in. outer and 0.5905 in. inner diameter, heated in a suitable furnace. Into the tube is lowered by means of platinum wires a platinum cup, 0.5905 in. high, 0.3543 in. diameter, with walls 0.006 in. thick, to receive from 0.5 to 1.2 g. of metallic sulphate to be decomposed. A tube holding the thermo-couple is introduced into the substance. This is heated from room-temperature to 1,100° C. in about

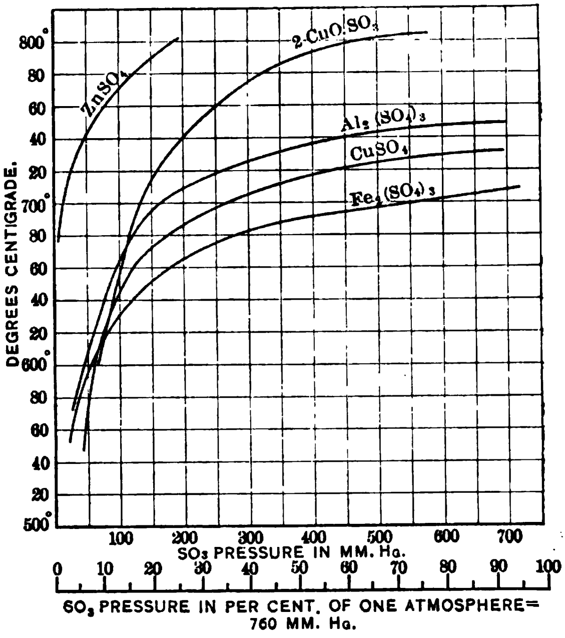

FIG. 1.—DECOMPOSITION OF SOME METALLIC SULPHATES IN AN EVACUATED TUBE.

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[3] *Metallurgie*, vol. vii., No. 11, p. 323 (June 8, 1910); *Stahl und Eisen*, vol. xxxi., No. 47, p. 1909 (Nov. 23, 1911).

1 hr.; the retardations due to absorption of heat by decomposition are noted. The results obtained are given in Table I.

TABLE I.—*Decomposition of Some Metallic Sulphates in a Tube Closed at One End.*

Metallic Sulphates.	Temperatures of Decomposition.	Products of Decomposition.	Transformations of Original Sulphates.	Fusions of Original Sulphates.
	Degrees C.		Degrees C.	Degrees C.
$Fe_2(SO_4)_3$	705	Fe_2O_3		
$CuSO_4$	740	$2CuO.SO_3$		
$Al_2(SO_4)_3$	770	Al_2O_3		
$NiSO_4$	840	NiO		
$ZnSO_4$	About 840	$3ZnO.2SO_3$	740	
$2CuO.SO_3$	845	CuO		
$CoSO_4$	880	CoO		
$3ZnO.2SO_3$	935	ZnO		
$MnSO_4$	1,030	?	860	700
CuO	About1,040	$(CuO.Cu_2O)$		
Ag_2SO_4	1,085	Ag	410	660
$PbSO_4$	....	....	850	

A comparison of the data with those obtained by other investigators will show that they are from 140° to 240° C. too high. The two leading reasons for this discrepancy are, that the decomposition takes place in an atmosphere of SO_3, respectively SO_2 and O, the partial pressures of which call for higher temperatures of dissociation than when heating in an air-current, as here the partial pressures of the products of decomposition are a negligible quantity; and, secondly, that the short time of 1 hr. given to each of the experiments must allow the temperature to rise above the critical point, considering the low conductivity for heat[4] of the sulphates. The temperatures obtained ought to correspond to those observed with an evacuated tube in which the pressure has reached 760 mm.; and this is approximately the case.

3. *Heating in a Crucible with Air-Current.*—The method has been followed by Bradford.[5] He obtained satisfactory results with the sulphates of iron, copper, and silver, by using a small quantity of substance and by placing the thermo-couple, in contact with substance, near to the crucible, as it is here that the dissociation begins. If much substance is used, or if the

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[4] H. Lescœur, *Comptes rendus de l'Académie des Sciences*, vol. cii., No. 25, p. 1466 (1886).

[5] *Trans.*, xxxiii., 58 (1902.

couple is placed in the center of the crucible, the temperature read for the beginning of a dissociation will be too high, on account of the low conductivity for heat of the substance.

4. *Heating in an Open Tube in a Current of Air.*—This method is the one which resembles more closely than any other the roasting-operation on a large scale. The substance to be tested is placed in a boat inserted into a horizontal tube impervious to gases; the thermo-couple is held near the boat or the substance; the tube is heated gradually, usually by electrical means; a slow currrent of purified and dried air is blown through and carries away the gaseous products of decomposition; the latter can be tested chemically whenever this seems desirable.

There are two ways of determining the beginning of a retardation caused by dissociation.

One is to heat to a given temperature, holding the thermocouple near the boat; to keep the temperature unchanged until any loss in weight of the substance has become constant, and to watch at the same time any change in the indicator, held in a railroad-tube, through which the gases have to pass. If a decomposition has taken place, another sample of the substance is treated in the same manner as the first, but to a lower temperature. In this way the range of temperature is narrowed down until the actual dissociation-temperature has been accurately ascertained. The loss in weight will show to what extent the substance has been decomposed; a chemical analysis may be necessary, if a chemical change other than expulsion of gaseous component has taken place. This method has been followed by Hofman with zinc sulphate,[6] Hofman-Mostowitsch with calcium sulphate,[7] Mostowitsch with barium sulphate,[8] Warlimont with iron, copper, and nickel sulphates,[9] Kothny with pyrite,[10] Landis with iron, copper, and zinc sulphates,[11] Mostowitsch with zinc sulphate,[12] and Barth with cobalt sulphate.[13] It has given accurate results as long as the necessary time has been given to the work.

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[6] *Trans.*, xxxv., 811 (1904).

[7] *Trans.*, xxxix., p. 628 (1908); xl., p. 807 (1909).

[8] *Metallurgie*, vol. vi., No. 14, p. 450 (July 22, 1909).

[9] *Metallurgie*, vol. vi., No. 3, p. 83 (Feb. 8, 1909).

[10] *Oesterreichisches Jahrbuch*, vol. lviii., pp. 97, 350 (1910); *Metallurgie*, vol. viii., No. 13, p. 389 (July 8, 1911).

[11] *Metallurgical and Chemical Engineering*, vol. viii., No. 1, p. 22 (Jan., 1910).

[12] *Metallurgie*, vol. viii., No. 24, p. 763 (Dec. 22, 1911).

The other method is to make first a blank test, using the amperage necessary to reach a desired temperature, with the thermo-couple held in the boat, but out of contact with the substance; to record the rise of temperature for given intervals of time, 30 or 60 sec., and to plot the data which will furnish a smooth curve. The next step is to charge the boat, raise the temperature in the same manner as with the empty boat, record the rise in temperature for the same intervals of time, and to plot the data. Any deviation of the second curve from the first will show that some change has taken place; there will have been a transformation if the weight of the substance has remained constant, a chemical change if the weight has increased or decreased. The method was devised by Wanjukow[14]; the abstract[15] contains some errors in his investigations upon cupric sulphate (see p. 914). As it is the method followed in the present research, the following example is given with some detail to make clear the mode of operation.

TABLE II.—*Dehydration of* $FeSO_4 + 7H_2O$.

CURRENT: 7 AMPERES AT 110 VOLTS.

Time-Interval.	Galvanometer Deflection		Time-Interval.	Galvanometer Deflection.		Time-Interval.	Galvanometer Deflection.	
	Blank Test.	Test With $FeSO_4$ + $7H_2O$.		Blank Test.	Test With $FeSO_4$ + $7H_2O$.		Blank Test.	Test With $FeSO_4$ + $7H_2O$.
Sec.	Millivolt.		Sec.	Millivolt.		Sec.	Millivolts.	
60	....	0.13	60	0.52	0.47	60	0.85	0.84
60	0.15	0.145	60	0.54	0.52	60	0.875	0.87
60	0.16	0.15	60	0.56	0.55	60	0.88	0.88
60	0.17	0.165	60	0.58	0.58	60	0.90	0.90
60	0.18	0.170	60	0.60	0.60	60	0.91	0.91
60	0.20	0.180	60	0.62	0.60	60	0.92	0.91
60	0.22	0.215	60	0.64	0.61	60	0.935	0.91
60	0.24	0.24	60	0.66	0.62	60	0.95	0.92
60	0.27	0.265	60	0.675	0.63	60	0.96	0.95
60	0.30	0.295	60	0.69	0.65	60	0.97	0.97
60	0.325	0.32	60	0.715	0.66	60	0.985	0.98
60	0.35	0.345	60	0.735	0.68	60	1.00	1.00
60	0.37	0.365	60	0.75	0.71	60	1.01	1.01
60	0.39	0.38	60	0.77	0.73	60	1.02	1.02
60	0.42	0.39	60	0.785	0.75	60	1.035	1.03
60	0.45	0.41	60	0.805	0.775	60	1.05	1.05
60	0.48	0.44	60	0.82	0.80	60	1.06	1.06
60	0.50	0.45	60	0.84	0.82	60	1.07	1.07

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[13] *Metallurgie*, vol. ix., No. 6, p. 199 (Mar. 22, 1912).

[14] *Journal der russischen physikalisch-chemischen Gesellschaft*, vol. xli., p. 680 (1909).

[15] *Chemisches Centralblatt*, vol. lxxx., p. 1124 (1908).

Table II. gives the observations made with the dehydration of green vitriol, $FeSO_4 + 7H_2O$. Column 1 shows that a record was taken every 60 sec.; in column 2 are noted the observations made with the empty boat; and in column 3 those with the boat charged with 0.3 g. of sulphate. A graphical representation of the data is given in Fig. 2, in which the ordinate gives the millivolts observed and the abscissa the time in minutes. The upper smooth curve, full-drawn, represents the blank test. In the lower curve the dotted line is a copy of the curve of the blank test; the full-drawn line gives the observations made with the charged boat; the numerical data added to the full-drawn line represent the degrees centigrade scale corresponding to the readings on the millivoltmeter. Following the pair of curves, they coincide until the temperature of 20° C. has been reached, when a decided retardation takes place, which lasts until the temperature has risen to 72° C. Weighings of the boat show that in the interval between 20° and 72° C. 3 molecules of water have been expelled. The weight of the charged boat now remains constant between 72° and 80° C. Here a second retardation is noticed, which extends from 80° to 123° C., and corresponds to an additional loss of 3 molecules of water, proved by the difference in weight. Between 123° and 132° the line curves coincide; if not theoreti-

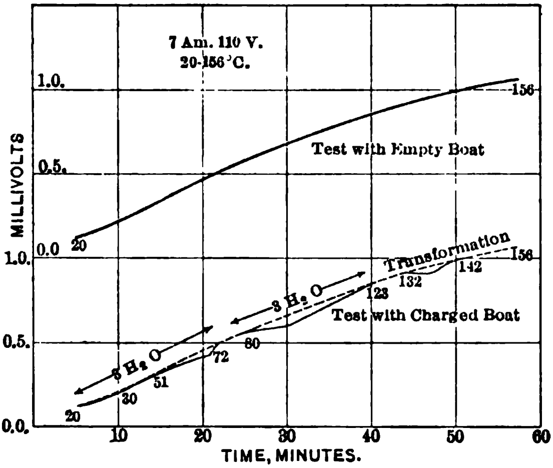

FIG. 2.—DEHYDRATION OF $FeSO_4 + 7H_2O$ IN AN OPEN TUBE WITH AIR-CURRENT.

cally, they do it at least practically. In the interval between 132° and 142° there is no change in weight; the retardation indicates a transformation, which is shown by a change in the color of the salt from white to gray. The expulsion of the last molecule of water is accompanied by the formation of a basic ferric salt, discussed on page 900.

II. PLAN OF INVESTIGATION.

The original plan of the investigation was, to determine the dissociation-temperatures of anhydrous metallic sulphates, the formation of basic sulphates, the characters of the resulting products, and the speeds of reaction at different temperatures, in order to obtain data applicable to roasting-operations. As the start had to be made in most cases with the chemically pure hydrous sulphates, the original plan was extended to include the dehydration of the hydrous salts by heating in a current of air. This research, besides having a general interest, has the practical value of determining (1) the temperatures at which sulphates with given amounts of water of crystallization can separate from their solutions, and (2) the temperatures to which crystals may be warmed in artificial drying without losing some water of crystallization. Thus, in order to obtain crystals of blue vitriol of the composition $CuSO_4 + 5aq$, the crystallization may not begin before the solution has cooled down to below 28° C., as otherwise $CuSO_4 + 3aq$ would separate, because at 28° C., $CuSO_4 + 5aq$ gives up 2 molecules of water; for the same reason the drying of crystals of $CuSO_4 + aq$ may not be carried on at a temperature above 28° C.

The method of heating in a current of purified and dried air was used in the dehydration-tests for the same reasons as those in desulphatization.

Two other methods have been used by early investigators. Thus Debray[16] applied the vacuum method discussed on page 890; Müller-Erzbach,[17] what may be called the evaporation method. In the latter the vapor-tension of a saturated salt-solution and of water are compared by placing the two in same-size tubes in a desiccator charged with concentrated sulphuric acid. The loss of water in the two cases gives the ratio of vapor-pressures. The ratio remains constant as long as the original salt solution remains unaltered, it changes as soon as a new hydrate begins to be formed. Thus 2 g. of sodium phosphate, $Na_2H.PO_4 + 12aq$, lose at first 5 molecules of H_2O, then again 5, and lastly 2 molecules, proving thus the existence of three hydrous compounds.

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[16] *Comptes rendus de l'Académie des Sciences*, vol. lxvi., No. 4, p. 194 (1868).

[17] *Annalen der Physik und Chemie*, vol. xxiii., p. 617 (1884).

With the exception of the sulphates of copper, calcium, and barium, all the sulphates of any metallurgical importance have been investigated, and the new results compared with the earlier. The three sulphates first mentioned have been accepted, as they were carried out under similar conditions by us and a former associate. The paper concludes with a general review of the subject and two final tables of data, which we consider reliable for dehydration and desulphatization.

III. HEATING-, MEASURING-, AND TESTING-APPARATUS.

The furnace used for heating was an electric resistance tube-furnace similar to the well-known Heraeus furnace employed by Hofman and Mostowitsch [18] in their work upon calcium sulphate, but contained a few improvements. Figs. 3 and 4 give a longitudinal section and an end-view. The heating-tube, *A*, of unglazed Marquardt material, 1.968 in. in diameter and 11.81 in. long, is wound with platinum foil; it is contained in a fire-clay tube, *B*, 3.543 in. inner diameter, and centered in it by fire-clay rings, *C*, at the ends, leaving an air-space, *D*, 0.6299 in. wide. The fire-clay tube is surrounded by calcined magnesia, *E*, inclosed in a sheet-iron cylindrical casing, *F*, 10.68 in. in diameter, with one flanged movable end, *G*, secured by screw-bolts, *H*, and thumb-nuts, *I;* the bolts are flattened at one end and riveted to the casing. The narrow annular space between casing *F* and tube *B* is closed with strips of asbestos paper soaked in starch-paste. The clay rings *C* are held in place by two triangular pieces of sheet-iron, *J*, backed by asbestos board, *K*, and attached to casing *F* by three threaded bolts, *L*, and nuts *M*. The silica tube, *N*, 0.7874 in. in diameter and 19.68 in. long, lies in the center of the heating-tube *A;* the small open annular spaces in rings *C* are closed with a clay-cement plaster. The tube holds the porcelain boat, *O*, 1.575 in. long by 0.2756 in. wide by 0.1968 in. deep, which contains the substance to be tested; both ends of tube *N* are closed with perforated rubber stoppers, *P*; at the opening *Q* is inserted a ⊥ glass-tube for introducing the thermo-electric pyrometer, *R*, with protection

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[18] *Trans.*, xxxix., 631 (1908).

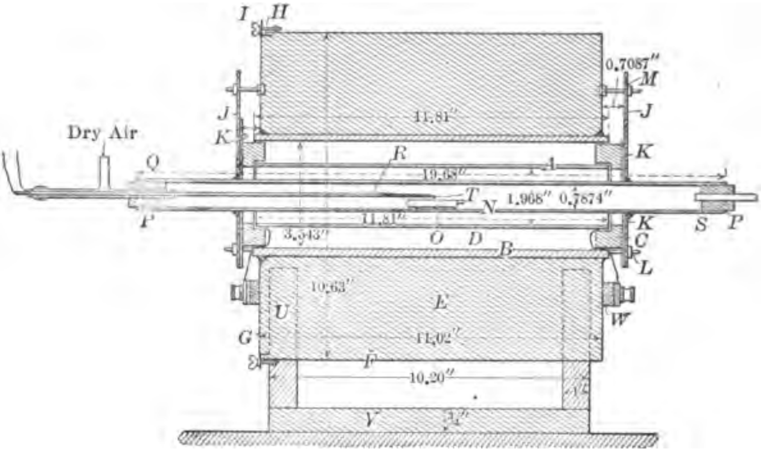

Fig. 3.—Electric Resistance Tube-Furnace. Longitudinal Section.

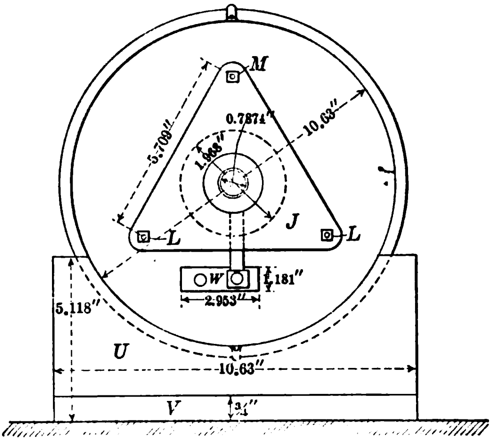

Fig. 4.—Electric Resistance Tube-Furnace. End-Elevation.

silica tube, 0.1181 in. in diameter, isolating silica tube, 0.0295 in. in diameter, and 0.0098-in. walls, and admitting air under pressure; at the opening *S* is a glass tube to be connected with a railroad-tube holding the indicator. The hot junction, *T*, of the couple is uncovered. The furnace rests in semi-circular recesses of two pieces of asbestos board, *U*, 1 in. thick, nailed to board, *V*, of the same material, 0.75 in. thick. Board *U* holds the binding-posts, *W*, for the electric connection.

During a test the uncovered thermo-couple was held above the substance; the electric measurements were made with a Siemens-Halske millivoltmeter having 180 decimillivolt divisions for 1,700° C.; the cold-junction was held in ice-water; the couple was standardized against the boiling-points of water, naphthaline and sulphur, and the freezing-points of water, aluminum, and the eutectic copper-cuprous oxide.

The current used was of 110 volts; it passed through the ammeter and two resistance-coils, one for rough and the other for close regulation. With all the resistance turned on, the current was of 2 amperes; with all turned off, the strength was 18 amperes, and the latter sufficed to attain 1,150° C.

It had been the intention to use a self-regulating rheostat, which was to consist of a non-conducting cylinder covered with resistance-wire, a gliding contact, and a clock to rotate the cylinder. This would have given straight lines instead of the curved ones shown in the records, but, unfortunately, the apparatus could not be made ready in time for this work. During our experiments an apparatus aiming at the same result, but of different construction, was described by Friedrich.[19]

In an experiment a certain number of amperes were made to pass through the winding of the furnace, and kept constant; records were taken every 60 sec.; the boat was removed from the furnace, transferred to a desiccator, and weighed; the substance was simply examined, or tested, as may have been found necessary.

Air under pressure was obtained by means of a Beutell apparatus connected with the water-service of the laboratory; the velocity was regulated by means of screw-clamps to furnish in the wash-bottles from 45 to 50 bubbles per minute. The air, before entering the furnace, was purified by passing it through wash-bottles containing a $\frac{1}{10}$ normal solution of potassium permanganate, concentrated potassium hydroxide, concentrated barium hydroxide, concentrated sulphuric acid, and a dryingtower, 15 in. high and 1.75 in. in diameter, charged with calcium chloride and phosphor pentoxide.

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[19] *Stahl und Eisen*, vol. xxxi., No. 50, p. 2045 (Dec. 14, 1911).

The gases leaving the furnace were passed through a colorless solution of potassium iodate and starch[30] for the detection of SO_2, and through a solution of barium chloride, acidulated with hydrochloric acid, to test for SO_3.

In making dehydration-tests at temperatures not exceeding 120° C., the preliminary indications of the electric resistance furnace were followed up with a Reichert thermostat, provided with the gas-pressure regulator of Bausch & Lomb. This enabled us to hold temperatures for any length of time within from 1° to 2° C.

IV. Record of Results.

1. *Ferrous Sulphate*, $FeSO_4 + 7H_2O$.

This salt forms bluish-green monoclinic crystals, which weather on exposure to air, being superficially oxidized to a basic ferric sulphate. Besides the common salt, there exist[31] the compounds $FeSO_4 + 4H_2O$, $FeSO_4 + 1H_2O$, and $FeSO_4$. The salt $FeSO_4 + 7H_2O$ is said to lose[32] 6 molecules of H_2O at 115° and to retain the 7th at 280°. Heating with exclusion of air gives[33] white $FeSO_4$, while with access of air[34] the basic ferric sulphate, $Fe_2O_3.2SO_3$, is formed. Several tests have recently been made upon the behavior of $FeSO_4$ when heated in a current of air. Warlimont[35] found that dissociation began at 470° and was complete at 550°. Bradford[36] noted the first retardation at 540° and gives 550° as the temperature at which the salt is completely decomposed. Friedrich[37] noticed the first absorption of heat at 700° and gives the dissociation-temperature as 705°. Kothny[38] finally found that at 150° $FeSO_4$ begins to be oxidized without any expulsion of SO_2, that at

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[30] Hofman, *Trans.*, xxxv., 817 (1904).

[31] Müller-Erzbach, *Annalen der Physik und Chemie*, vol. xxvi., p. 409 (1885).

[32] Graham, *Philosophical Magazine*, Third Series, vol. vi., No. 36, p. 421 (June, 1835).

[33] Brandes, *Schweigger's Journal*, vol. li., p. 438 (1827).

[34] Waltl, *Chemisches Repertorium für Pharmacie*, vol. xli., p. 428 (1902).

[35] *Metallurgie*, vol. vi., No. 4, p. 131 (Feb. 22, 1909).

[36] *Trans.*, xxxiii., 50 (1902).

[37] *Metallurgie*, vol. vi., p. 132 (1909).

[38] *Oesterreichisches Jahrbuch*, vol. lviii., p. 115 (1910).

480° it is completely converted into $Fe_2O_3.2SO_3$, with a loss of only 4.2 per cent. of the S in the form of SO_2, and that at 530° the basic salt is completely dissociated into Fe_2O_3 and SO_3, but very slowly.

A chemical analysis of the salt used in the tests is given in Table III.

TABLE III.—*Composition of* $FeSO_4 + 7H_2O$.

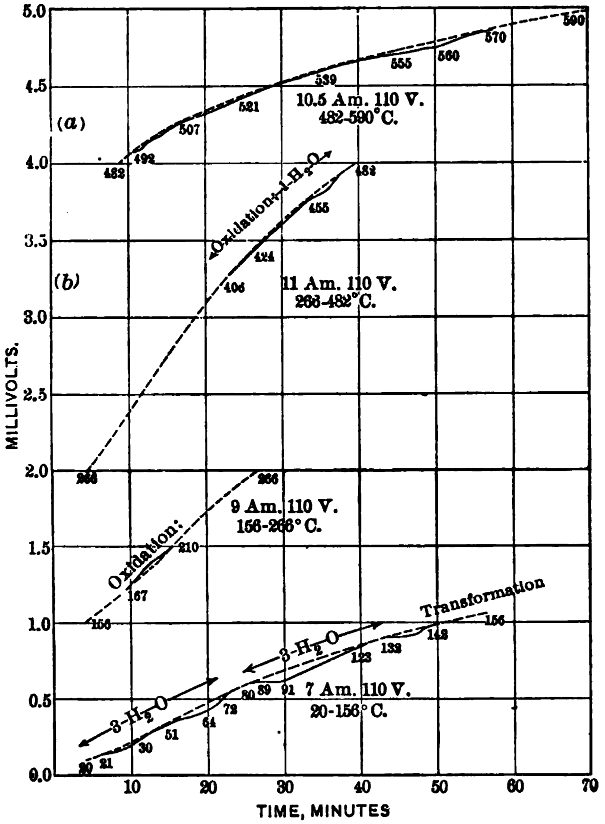
Component.	Chemical Analysis.	Calculated Analysis.
	Per Cent.	Per Cent.
SO_3	28.82	28.77
FeO	26.01	25.91
H_2O	45.17[a]	45.32
SO_3 . FeO	1.1080	1.1103

[a] By difference.

The iron was precipitated, after oxidation with nitric acid, from a boiling solution with ammonia, and weighed after ignition as Fe_2O_3. The sulphur was determined as $BaSO_4$, from a chloride solution, by precipitating with barium chloride.

The substance used in the test weighed 0.3 g., and the charged boat 4.1672 g. The bottom curve in Fig. 5 represents the heating from 20° to 156°. At 21° occurs the first retardation, which lasts until the temperature has risen to 30°; it begins again at 51°, and terminates at 72°. Parallel tests in the thermostat showed that heating to 25° for 8 hr. expelled 3 molecules of H_2O. At 80° there is a second deviation from the curve of the blank test, which disappears at 123°; again 3 molecules of H_2O have been drawn off. Test with the thermostat showed that heating to 120° for 25 hr. removed 3 molecules of H_2O. The absorption of heat lasting from 132° to 142° corresponds to a transformation; the substance, originally white, has become gray, and has not undergone any change in weight. The weight of the charge was 4.0568 g. after the temperature had reached 156°; further heating for 1.75 hr. at this temperature reduced it to 4.0547 g., and for an additional hour to 4.0527 g. This corresponds to a total reduction of 0.1145 g., which is equal to 6 molecules of H_2O. Between 156°

and 266° the curve for the charged boat rises above that for the blank test, on account of the heat set free by the oxidation of the ferrous salt. The boat, removed after 266° had been reached, weighed 4.0534 g.; exposing it for 0.5 hr. more to 266° increased the weight only to 4.0535 g. The substance, gray after the transformation, had become a greenish yellow; the formation of a basic salt had begun, but the 7th molecule of



FIGS. 5a AND 5b.—FERROUS SULPHATE, $FeSO_4 + 7H_2O$.

H_2O had not yet been expelled. Between 266° and 482° no change occurs up to 406°, when there is a slight deviation for the curve of the charged boat from that of the blank test, reaching to 455°, when the absorption of heat becomes more marked; now the boat weighed 4.0484 g. Returning the boat to the furnace cooled to 475°, heating for 20 min. reduced the weight to 4.0466 g., for 30 min. more to 4.0418 g., and again for 30 min. to 4.0420 g. The substance has become brownish-yellow. Two processes have taken place, the ferrous iron has been oxidized, and the 7th molecule of H_2O has been expelled. The total loss in weight was 0.1252 g., *i. e.*, 0.0107 g. less than called for by the 7 molecules of H_2O (= 0.1859 g.); the difference is due to the oxidation of the ferrous iron, accompanied by the formation of Kothny's $Fe_2O_3.2SO_3$.

The curve in Fig. 5*a* extending from 482° to 590° shows at 492° the first retardation, due to the beginning of the dissociation of the $Fe_2O_3.2SO_3$. The process proceeds very slowly and becomes marked only between 550° and 560°. This range of temperature can, therefore, be taken as the practical temperature for the decomposition of $FeSO_4$, which, however, is noticeable already at 492°.

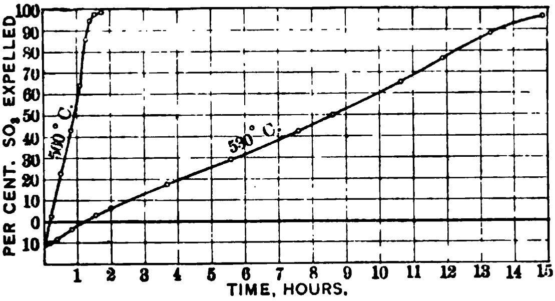


FIG. 6.—VELOCITY OF DECOMPOSITION OF $FeSO_4$ IN A CURRENT OF AIR AT 530° AND 560° C.

The velocity of decomposition at 530° and 560° C. is shown in Fig. 6.

There exist three hydrous salts, $FeSO_4 + 7H_2O$; $FeSO_4 + 4H_2O$; $FeSO_4 + 1H_2O$. The salt $FeSO_4 + 7H_2O$ gives up 3 molecules of H_2O at 21°, 3 more at 80°, undergoes a transformation at 132°, begins to be oxidized at 167°, but very slowly; the oxidation continues with the rise of temperature to 455°, when the oxidation to $Fe_2O_3.2SO_3$ is completed, and at the same time the last molecule of H_2O is expelled. The dissociation of $Fe_2O_3.2SO_3$ begins at 492° and is finished at from 555° to 560°.

2. *Bismuth Sulphate, $Bi_2(SO_4)_3$.*

The normal sulphate is a white powder which readily combines with H_2O to form $2Bi_2(SO_4)_3 + 7H_2O$.[29] The decomposition of the anhydrous salt[30] begins between 405° and 418°; at a higher temperature $Bi_2O_3.SO_3$ is formed[31]; upon further increasing the heat the dissociation is complete,[32] but, according to Schmidt,[33] is accompanied by the volatilization of some Bi_2O_3. Abegg[34] doubts the existence of basic sulphates.

The normal salt used in the experiments contained 5 per cent. of H_2O, as shown in Table IV. It was treated with water and some ammonium sesquicarbonate was added, the resulting basic bismuth carbonate was filtered, dried, fused in a porcelain chamber with five times its weight of 98-per cent. potassium cyanide, and the resulting metal weighed. The sulphur was determined by precipitating with barium chloride from a chloride solution and weighing as $BaSO_4$. The results are given in Table IV.

TABLE IV.—*Composition of* $Bi_2(SO_4)_3$.

Component.	Chemical Analysis.	Calculated Analysis.
	Per Cent.	Per Cent.
SO_3	37.75	39.735
Bi_2O_3	57.25	60.265
H_2O	5.00[a]	none
$SO_3 : Bi_2O_3$	0.6595	0.6594

[a] By difference.

The substance tested weighed 0.3 g.; the weight of the charged boat was 5.0144 g. Heating to 170° reduced the weight to 4.9994 g. by the expulsion of 0.0150 g. of H_2O. No retardations were noted in the curve, Fig. 7*a*, while heating from 170° to 347°, and from 347° to 495°; the indicator remained uncolored, and the weight of the boat unchanged. The salt was light yellow when hot, but became again white when cold.

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[29] Gmelin-Kraut, *Handbuch der anorganischen Chemie*, vol. iii., part ii., p. 981 (1908).

[30] Bailey, *Journal of the Chemical Society*, vol. li., p. 676 (1887).

[31] Heintz, *Annalen der Physik und Chemie*, vol. lxiii., p. 77 (1844).

[32] Bailey, *loc. cit.*

[33] *Berichte der deutschen chemischen Gesellschaft*, vol. xxvii., pt. 1, No. 2, p. 236 (1894).

[34] *Handbuch der anorganischen Chemie*, vol. iii., part ii., p. 665 (1907).

In the curve representing the rise of temperature from 495° to 666°, Fig. 7*b*, a retardation is seen at 570°, which is due to decomposition setting in; this progresses very slowly to 603°, then becomes more noticeable up to 639°, when occurs a second decided retardation, explained by the formation of a basic salt, yellow when hot, though less so than the normal salt, and white when cold. In order to ascertain the composition of the basic salt, 0.3 g. of substance was weighed out and brought to constant weight at 630°. The charged boat weighed 5.0144 g.; heating to 170° expelled the hygroscopic water and reduced the weight to 4.9994 g. Heating at 630° for five 20-min. periods gave the weights 4.9832, 4.9804, 4.9770, 4.9740, and 4.9744 g.; that is, a loss of 0.0250 g., referred to the weight of dry substance. This loss corresponds to one-fifth of the total

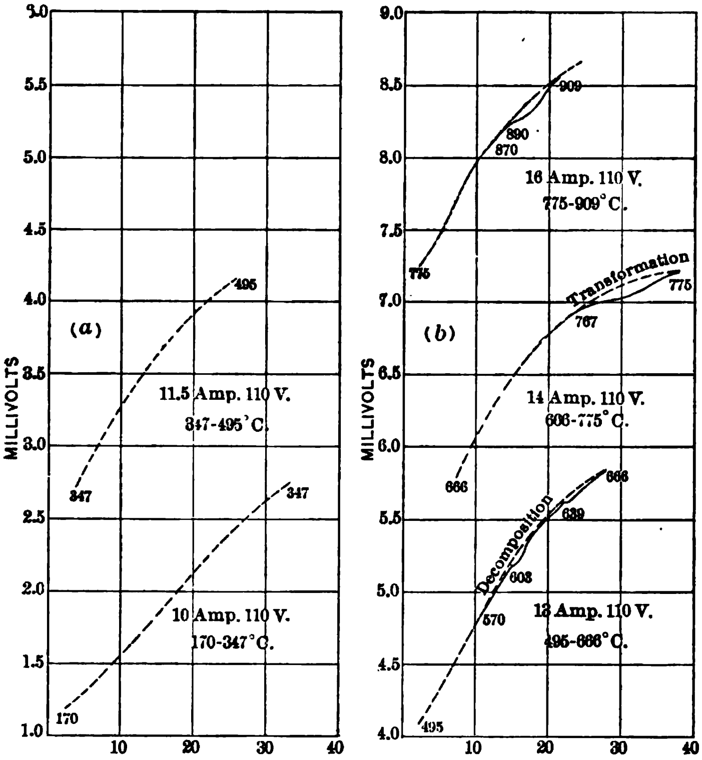

Fig. 7*a*. Fig. 7*b*.

FIGS. 7*a* AND 7*b*.—DECOMPOSITION OF $Bi_2(SO_4)_3$.

SO_3, and gives the basic salt the formula $5Bi_2O_3.4(SO_3)_3$. This is white, both hot and cold, and begins to form already at 570°. The curve giving temperatures ranging from 666° to 775° has a retardation, which begins at 767° and corresponds to a transformation. The salt has not changed either in weight or in color, but has shrunk greatly in volume. In heating from 775° to 909° there is noted an absorption of heat at 870°, accompanied by a coloration of the indicator, which shows the beginning of the dissociation of $5Bi_2O_3.4(SO_3)_3$; this is very decided at 890°, but not yet complete. At 922° (not shown) another retardation was seen, and this was due to fusion of the partly decomposed basic salt. Heating the fused salt to 900° for 4.5 hr. and to 1,020° for 1 hr. gave a diminution in weight of 0.0280 g., due in part to dissociation of the basic salt, in part to volatilization.

The normal anhydrous salt, $Bi_2(SO_4)_3$, begins to be converted at 570° into $5Bi_2O_3.4(SO_3)_3$, but the change progresses slowly; the basic salt undergoes a transformation at 767°, and a partial decomposition at 870°, which is not completed when the salt fuses at 922° and is further heated for a long time at 1,020° with partial volatilization.

3. *Aluminum Sulphate,* $Al_2(SO_4)_3 + 16H_2O$.

Pure alum crystallizes with 16 molecules of H_2O, forming a stable compound; the impure salt often crystallizes with 18 molecules of H_2O and is hygroscopic.[35] Upon heating, the hydrous salt is said to fuse in its water of crystallization and to be decomposed without forming a basic salt.[36] Pickering[37] found that the so-called basic salts are mixtures of Al_2O_3 and $Al_2(SO_4)_3$. Wöhler-Plüddemann-Wöhler[38] determined the dissociation-temperature of the anhydrous salt at 750°; Friedrich[39] noticed retardations at 100° and 350°, and fixed the decomposition at 770°.

The composition of the salt used in the experiments is given in Table V.

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[35] Mendelejeff, *Grundlagen der Chemie*, p. 751 (1891).

[36] Dammer, *Handbuch der anorganischen Chemie*, vol. iii., p. 98 (1893.)

[37] *Chemical News*, vol. xlv., pp. 121, 133 (Mar. 24, 31, 1882).

[38] *Berichte der deutschen chemischen Gesellschaft*, vol. xli., pt. 1, No. 4, p. 703 (1908).

[39] *Metallurgie*, vol. vii., No. 11, p. 333 (June 8, 1910).

TABLE V.—*Composition of* $Al_2(SO_4)_3 + 16H_2O$.

Component.	Chemical Analysis.	Calculated Analysis.
	Per Cent.	Per Cent.
SO_3	38.08	38.081
Al_2O_3	16.12	16.22
H_2O	45.80 [a]	45.699
$SO_3 : Al_2O_3$	2.3635	2.3479

[a] By difference.

The alumina was precipitated with ammonia and weighed as Al_2O_3; the sulphur was separated from a chloride solution with barium chloride and weighed as $BaSO_4$.

For the experiments 0.3 g. was taken; the charged boat weighed 4.1100 g. The bottom curve in Fig. 8*a* represents heatings from 15° to 130°. The first retardation was noticed at 51° and extended to 80°, the second lasted from 82° to 97°, the third from 97° to 105°, and the fourth from 109° to 130°. Each of the first three retardations is caused by an expulsion of 3 molecules of H_2O. Parallel tests in the thermostat showed that heating at 80° for 30 hr. eliminated 3 molecules of H_2O, at 87° for 6.5 hr. again 3 molecules, at 105° for 5.5 hr. 3 more molecules, and at 170° for 33 hr. again 3 molecules of H_2O. The long time required to remove the last 3 molecules finds a parallel in the fact that in the tube-furnace at 130° only part of these 3 molecules had been expelled, the time given being too short. The charged boat heated to 130° weighed 4.0230 g.; the loss in weight had been 0.0870 g. instead of 0.1027 g. In the second curve, covering from 130° to 276°, the temperature has risen to 180° before the 12th molecule was completely expelled. Three more molecules of H_2O are driven out between 180° and 263°. The boat, after being held for 30 min. at 263°, weighed 3.9938 g.; for an additional 60 min., 3.9866 g.; and for 30 min. more, 3.9860. g. It took 120 min. at 263° to expel the 15th molecule of H_2O, which shows the tenacity with which the water is held. The next two curves, covering 276° to 380° and 380° to 544°, Figs. 8*a* and 8*b*, show the elimination of the 16th and last molecule of H_2O, which starts at 316° and is finished at 442°. The weight of the charged boat had been reduced to 3.9732 g., which corresponds to a loss of 0.1368 g., while the theoretical figure calls for 0.1363 g. The salt was snow-white. In the last curve, covering from 544° to 746°, the dissociation of the anhydrous salt is seen to begin at 590°; it progresses slowly, becomes marked at 639°, grows weaker, and is complete at 690°. The charged

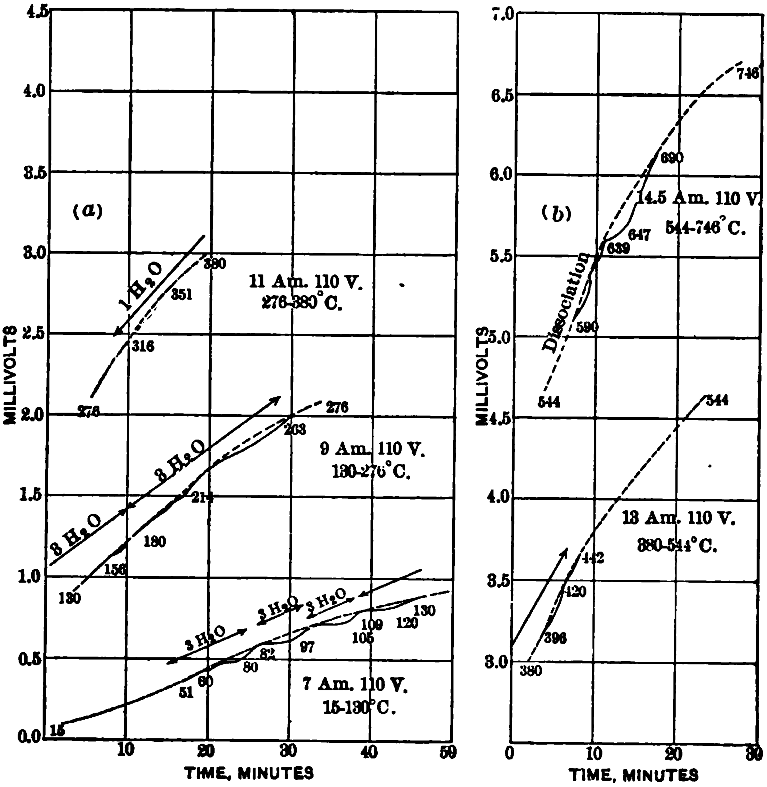

Fig. 8a. Fig. 8b.

Figs. 8a and 8b.—Dehydration and Desulphatization of $Al_2(SO_4)_3 + 16H_2O$.

boat, removed at 746°, weighed 3.9545 g., which is equivalent to a loss of 0.0187 g. and corresponds to the SO_3 given off. The velocity of decomposition at 755° is represented graphically in Fig. 9.

There exist six hydrous salts: $Al_2(SO_4)_3 + 16H_2O$; $Al_2(SO_4)_3 + 13H_2O$; $Al_2(SO_4)_3 + 10H_2O$; $Al_2(SO_4)_3 + 7H_2O$; $Al_2(SO_4)_3 + 4H_2O$; $Al_2(SO_4)_3 + 1H_2O$.

The salt $Al_2(SO_4)_3 + 16H_2O$ gives off 3 molecules of H_2O at 51°, 3 at 82°, 3 at 97°, 3 at 109°, 3 at 180°, and the last one at 316°. The dissociation of the anhydrous salt begins at 590° and becomes marked at 639°.

4. *Lead Sulphate, $PbSO_4$.*

Anglesite forms orthorhombic crystals, the prepared sulphate a white crystalline powder.

In a general way it is known that this salt loses some of its SO_3 at elevated temperatures. Doeltz and Graumann[40] found that in heating in an air-current for 30-min. periods the salt suffered no change in weight up to 800°, that at 900° a loss of from 0.32 to 0.42 per cent. took place, that at 1,000° much SO_3 was expelled, causing in 1.5 hr. a reduction in weight of 14.3 per cent. Ramsay and Eumorfopoulos[41] determined the melting-point of $PbSO_4$ as 937°, which according to the research of Schenck and Rassbach (see below) must have been that of the partly decomposed salt. Boussingault[42] claims to have decomposed the salt completely. Berthies[43] prepared the compound $2PbO.SO_3$ by fusion of $PbSO_4$ and PbO; he says it is a light-yellow substance when hot, and white when cold. Schenck and

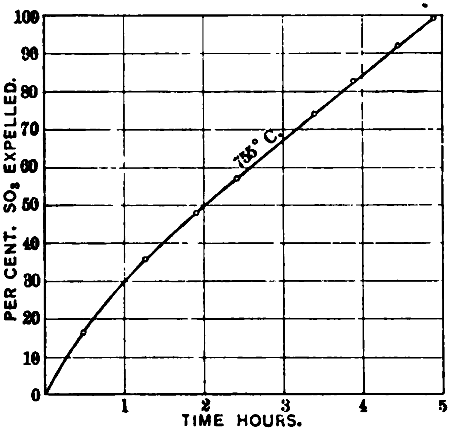

Fig. 9.—Velocity of Decomposition in a Current of Air of $Al_2(SO_4)_3$ at 755° C.

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[40] *Metallurgie*, vol. iii., No. 13, p. 441 (July 8, 1906).

[41] *Philosophical Magazine*, Fifth Series, vol. xli., No. 251, p. 360 (Apr., 1896).

[42] *Annales de Chimie et de Physique*, Fourth Series, vol. xii., p. 246 (1867).

[43] *Op. cit.*, vol. xliii., p. 217 (1830).

Rassbach[44] have traced the freezing-point curve of the series $PbSO_4$—PbO. They found that the melting-point of $PbSO_4$ lay above 1,100°, and that at 950° a retardation occurred, which was caused by a partial decomposition. Their curve shows the existence of two stable basic sulphates: $2PbO.SO_3$, with melting-point of 960°, and $3PbO.SO_3$, with melting-point of 951°; a third basic salt, $4PbO.SO_3$, is stable only below 880°, as above this temperature it splits into $3PbO.SO_3$ and PbO. They noted also two transformations, $PbSO_4$ at 850°, and $3PbO.SO_3$ at 450°.

An analysis of the lead sulphate used in the experiments gave the results shown in Table VI.

TABLE VI.—*Composition of $PbSO_4$.*

Component.	Chemical Analysis.	Calculated Analysis.
	Per Cent.	Per Cent.
SO_3	26.13	26.43
PbO	73.45	73.57
SO_3 : PbO	0.3557	0.3592

For the experiment there was weighed out 0.3 g. of substance; the charged boat weighed 4.1690 g. Heating to 181° reduced the weight to 4.1682 g. by the expulsion of water. The curves, Fig. 10*a*, covering 181° to 297° and 297° to 553°, show slight deviations for the charged boat between 195° and 238°, and between 345° and 418°; both are accompanied by bluing of the indicator, showing that some SO_2 is set free. The loss in weight caused by the first retardation was 0.0012 g., that by the second 0.0080 g., or 2.66 per cent. The white color of the original substance had changed to gray. Heating for an additional 20 min. at 553° caused no further loss in weight; it did, however, restore the original white color. In the temperature-range between 553° and 770°, Fig. 10*b*, retardations are noticeable at 637° and 705°; both are again accompanied by a slight setting-free of SO_2. The boat heated to 750° weighed 4.1566 g.; heating for one more hour at this temperature reduced the weight to 4.1560 g. The total loss in weight endured so far was 0.0130 g., which corresponds to one-sixth of the SO_3 of 0.3 g. of substance. In order to verify these observations, again 0.3 g. of substance was taken (weight of charged boat was 4.3402 g.) and heated to 750°. After 4 or 5 min. the bluing of the indicator showed that SO_2 was

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[44] *Berichte der deutschen chemischen Gesellschaft,* vol. xli., pt. 2, No. 13, p. 2917 (1908).

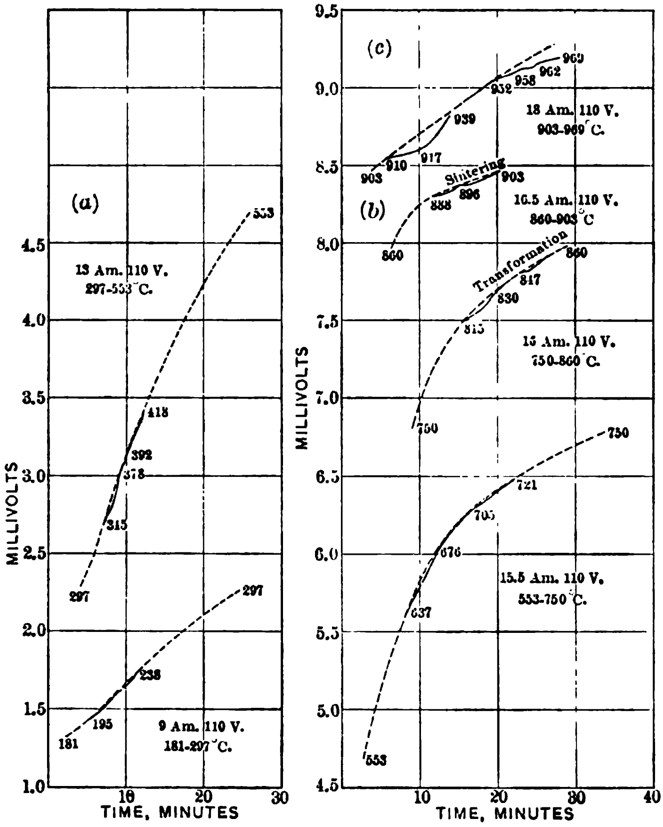

Fig. 10a.

Figs. 10*b* and 10*c*.

FIGS. 10*a*, 10*b*, AND 10*c*.—DESULPHATIZATION OF $PbSO_4$.

being liberated. After 35 min. at 750° the weight of the charged boat was 4.3283 g., after 30 min. more 4.3276 g., after an additional 20 min. 4.3276 g.; that is, no more loss took place. The total loss was equal to one-sixth of the SO_3 present, hence the formula for the newly-found basic salt is $6PbO.5SO_3$. Between 750° and 860° there is a small retardation at 815°, accompanied by the evolution of some SO_2, and a large one at 847°, the temperature remaining constant there for 1.5 min. The charged boat, taken out at 860°, weighed 4.3267 g., the slight loss being caused by the expulsion of some SO_3; there had, however, taken place a large shrinkage in the volume of the substance, showing that a transformation of the salt $6PbO.5SO_3$ had occurred, which is in agreement with the results of Schenck and Rassbach. Heating the charged boat for an additional 30 min. at 860° gave a weight of 4.3265 g., or a loss of only 0.0002 g. The curve 860°–903° shows two retardations, at 888° and 896°; the first is caused by the decomposition of $6PbO.5SO_3$, proved by the coloration of the indicator and by the loss in weight, the boat at 903° weighing 4.3236 g.; the second by the partial sintering of the salt. The last curve, Fig. 10*b*, giving the results obtained between 903° and 969°, shows marked differences between the curves for the charged and empty boats. At 910° the fusion of the salt begins and is completed at 939°; dissociation with evolution of SO_3 begins at 952°; at 958° and 962° retardations were noticed, which may correspond to Schenck and Rassbach's compounds $3PbO.SO_3$ and $2\ PbO.SO_3$. In order to test the expulsion of SO_3, again 0.3 g. of $PbSO_4$ was weighed out, the charged boat weighing 4.3402 g. Heating to 950° reduced the weight to 4.3265 g., holding at 950° for 8 hr. 10 min. diminished it to 4.2700 g., and for 20 min. more to 4.2698. Raising the temperature to 1,115° and holding the boat at this heat for 30 min. lowered the weight to 4.2596 g., holding it for 120 min. more gave 4.2580 g. The loss in weight of 0.0102 g., experienced between the last weight taken at 950° and the first at 1,115°, viz., 4.2698 and 4.2596, exceeded already the theoretically possible loss of SO_3 (*i. e.*, 0.0792 g.), and thus proved that the decomposition of the basic salt $2PbO.SO_3$ is accompanied by the volatilization of PbO.

The results show that the decomposition of $PbSO_4$ begins already at 195° but progresses very slowly, with the formation of the basic salt $6PbO.5SO_3$ at 705°; that this salt undergoes a transformation at 847°, begins to be decomposed at 888°, to sinter at 896°, and to fuse at 910°; rapid dissociation begins at 952°, and is accompanied by volatilization of PbO; retardations at 958° and 962° indicate an acceleration of the decompotion, which, however, is not complete.

5. *Cupric Sulphate,* $CuSO_4 + 5H_2O$.

Blue vitriol forms sky-blue triclinic crystals. The expulsion of its water of crystallization has been studied by Lescœur,[45] Andreae,[46] Müller-Erzbach,[47] and Wanjukow.[48] They are all in accord as to the existence of the following salts: $CuSO_4 + 5H_2O$; $CuSO_4 + 3H_2O$; $CuSO_4 + 1H_2O$; $CuSO_4$.

According to Magnier de la Source,[49] $CuSO_4 + 5H_2O$ is changed in a current of air of 25° to 30° into sky-blue $CuSO_4 + 3H_2O$. This has been re-investigated by Wanjukow, who fixed the range of conversion between 27° and 30°; he found also that the pale-blue $CuSO_4 + 1H_2O$ was formed between 93° and 99°. Graham[50] says that blue vitriol is completely dehydrated between 221° and 243°; Wanjukow found that the complete dehydration began already at 155° and was completed at 233°.

The decomposition of $CuSO_4$ by heating was probably first studied by Rousche,[51] who found that at a dark red the basic orange-colored salt $2CuO.SO_3$ was formed. In more recent times Bradford[52] found that in an air-current $CuSO_4$ was decomposed to $2CuO.SO_3$ at 653°, and the latter at 702° into CuO and SO_3.

Warlimont[53] ascertained that by heating for 3 hr. the decomposition of $CuSO_4$ to CuO and SO_3 began between 600° and 610°, and increased at 630° to 4.8 per cent., at 640° to 11.3 per cent., and at 650° to 100 per cent. Landis[54] noticed the first traces of SO_2 at 400°; a rapid evolution of gas took place at 690°; this diminished with rise of temperature until 900°

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[45] *Comptes rendus de l'Académie des Sciences,* vol. cii., No. 25, p. 1466 (1886).

[46] *Zeitschrift für physikalische Chemie,* vol. vii., p. 241 (1891).

[47] *Annalen der Physik und Chemie,* vol. xxvi., p. 409 (1885); *Zeitschrift für physikalische Chemie,* vol. xix, p. 144 (1896).

[48] *Journal der russischen physikalisch-chemischen Gesellschaft,* vol. xli., p. 710 (1909).

[49] *Comptes rendus de l'Académie des Sciences,* vol. lxxxiii., No. 20, p. 899 (1876).

[50] *Philosophical Magazine,* Third Series, vol. vi., No. 36, p. 418 (June, 1835).

[51] *Zeitschrift für Chemie und Pharmazie,* vol. iii., p. 462 (1860).

[52] *Trans.,* xxxiii., 59 (1902).

[53] *Metallurgie,* vol. vi., No. 4, p. 132 (Feb. 22, 1909).

[54] *Metallurgical and Chemical Engineering,* vol. vii., No 1, p. 327 (Jan., 1910).

was reached, when the dissociation was complete. Friedrich[55] found by his method that $2CuO.SO_3$ was formed at 700°, that this began to be decomposed between 840° and 850°, and that all sulphurous gas was expelled at 900°. Wanjukow[56] noticed retardations at 341°, 380°, 503°, and 621°, all accompanied by evolutions of SO_2 and the formation of increasing small amounts of the basic salt $8CuO.3SO_3$. Thus, at 341°, there was decomposed 0.51 per cent. of the $CuSO_4$; at 380°, 1.09 per cent.; at 503°, 2.38 per cent.; and at 621°, 5.87 per cent. In treating with water this $CuSO_4$, thus partly converted into $8CuO.3SO_3$, the $CuSO_4$ went quickly into solution, while the basic salt remained unchanged for 30 min.; boiling for 20 min. caused it to take up some water and form a pale-blue hydrous salt, which, however, was not dissolved. Heating the partly decomposed $CuSO_4$ to 653° caused the formation of the orange-colored basic salt $2CuO.SO_3$, which was slow up to 670°, when it progressed more rapidly. The expelled gas contained both SO_3 and SO_2. This salt did not change during an exposure to dry air for 4 months; treating it with water at ordinary temperature had no effect for 5 or 6 min., then gradually the salt took up some water and became bluish, but was not dissolved. Heating $2CuO.SO_3$, the first retardation was noticed at 704°, the decomposition to CuO became pronounced at 736°, and continued until the temperature had risen to 751°. At 787° the residue contained 0.1108 per cent. of SO_3, at 802° and 805° only a trace.

There exist three hydrous salts: $CuSO_4 + 5H_2O$; $CuSO_4 + 3H_2O$; and $CuSO_4 + 1H_2O$. The salt $CuSO_4 + 5H_2O$, upon heating in a current of dry air, is converted between 27° and 30° into $CuSO_4 + 3H_2O$, the latter at from 93° to 99° into $CuSO_4 + 1H_2O$, and the last at 155° into $CuSO_4$. The anhydrous $CuSO_4$ is converted at 341° into $8CuO.3SO_3$ to the limit of 5.87 per cent.; at 653° $2CuO.SO_3$ begins to be formed; at 704° the final dissociation into CuO is started; it becomes pronounced at 736°.

6. *Manganous Sulphate, $MnSO_4 + 5H_2O$.*

This salt forms peach-blossom monoclinic crystals, which slowly weather on exposure to air. Müller-Erzbach[57] recognizes three hydrous salts, with 5, 2, and 1 molecules of H_2O. Heating $MnSO_4 + 5H_2O$ to 150° expels 4 molecules of H_2O (Pettersson[58]), and to 280° causes the removal of all the H_2O (Thorpe-Watts[59]); Graham[60] gives from 187° to 200° as the critical temperature. According to Friedrich,[61] the expulsion of H_2O begins at 280°; he found that at 700° the anhydrous salt fused, and that at 1,030° dissociation took place.

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[55] *Metallurgie*, vol. vii., No. 11, p. 327 (June 8, 1910).

[56] *Journal der russischen physikalisch-chemischen Gesellschaft*, vol xli., p. 688 (1909).

The salt used in the experiments was somewhat weathered, as shown by the water-content in Table VII.

TABLE VII.—*Composition of* $MnSO_4 + 5H_2O$.

Component.	Chemical Analysis.	Calculated Analysis.
	Per Cent.	Per Cent.
SO_3	35.50	33.22
MnO	31.45	29.43
H_2O	33.05[a]	37.35
SO_3 : MnO	1.1287	1.1287

[a] By difference.

The manganese was determined by the Volhard method; the sulphate was precipitated from a chloride solution with barium chloride and weighed as $BaSO_4$.

For the experiment 0.3 g. of substance was taken; the charged boat weighed 4.1652 g. In Fig. 11*a*, the bottom curve gives the changes taking place between 22° and 115°. The first retardation is noticed at 25°; it extends to 60° and is accompanied by a loss of 3 molecules of H_2O. During the second retardation, lasting from 60° to 87°, another molecule of H_2O is expelled. Parallel tests in the thermostat showed that up to 25° no water was expelled, that heating to 30° for 44 hr. eliminated 3 molecules of H_2O, and to 100° for 25 hr. part of the 4th molecule, but that the temperature had to be raised to

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[57] *Annalen der Physik und Chemie,* vol. xxvi., p. 421 (1885).

[58] Gmelin-Kraut, *Handbuch der anorganischen Chemie,* vol. ii., part 2, p. 282 (1908).

[59] *Journal of the Chemical Society,* vol. xxxvii., p. 113 (1880).

[60] *Philosophical Magazine,* Third Series, vol. vi., No. 36, p. 420 (June, 1835).

[61] *Metallurgie,* vol. vii., No. 11, p. 329 (June 8, 1910).

120° to remove it completely. At 115° the charged boat weighed 4.0950 g., showing that a loss of 0.0702 g. had been experienced, while the expulsion of 4 molecules of H_2O raised the figure to 0.0741 g. The curve representing the range 115°–241° shows two retardations, at 148° and 187°, and the joining of the curves at 204°. The expulsion of the 5th molecule of H_2O begins at 152°. At 241° the charged boat weighed 4.0912 g.; holding it at this temperature for 1 hr. reduced the weight to 4.0716 g., and for 2 hr. more to 4.0676 g. The total loss was 0.0976 g.; the weight of 5 molecules of H_2O corresponds to 0.0971 g. The curves covering the ranges 241°–364°, and 364°–603°, Fig. 11*b*, show no thermal changes; the weight of the charged boat remained constant at 4.0676 g.

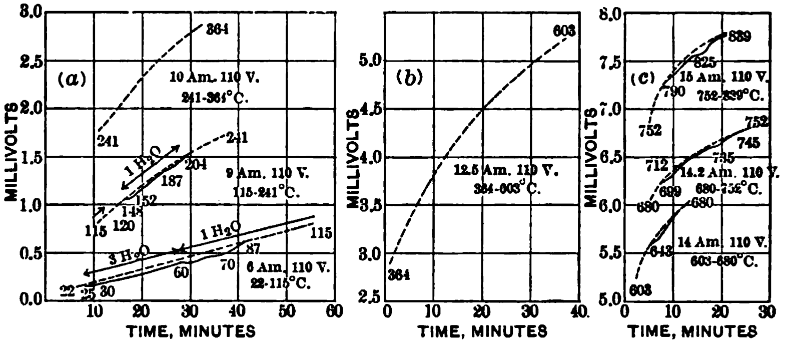

Fig. 11*a*. Fig. 11*b*. Fig. 11*c*.

FIGS. 11*a*, 11*b*, AND 11*c*.—DEHYDRATION AND DESULPHATIZATION OF $MnSO_4 + 5H_2O$.

The bottom curve of Fig. 11*c*, representing the heating from 603°–680°, shows an absorption of heat beginning with 643°, caused by a sintering of the edges of the crystals. Between 680° and 752° there is a decided retardation at 699°, which extends to 745°, and is caused by the dissociation of the anhydrous salt, the indicator showing the presence of SO_2 by turning blue. The salt upon examination was found to be not fused, and to have retained its pulverulent character. In order to fix more definitely the dissociation-temperature of $MnSO_4$, a new sample of 0.3 g. of substance was taken; the charged boat weighed 4.3310 g. Heating to 570° expelled all the water and reduced the weight to 4.2390 g. Holding it for

30 min. at 570° gave the weight 4.2384 g., the indicator showed the presence of SO_2; 30 min. at 700° did not change the weight, but the indicator began to show a bluish tinge; 30 min. at 710° reduced the weight to 4.2378 g. and decidedly colored the indicator, 30 min. at 730° gave 4.2372 g., 30 min. at 815° gave 4.2264 g., and the substance in the boat began to blacken. The last curve, representing heatings from 752° to 839°, has a clear retardation at 790°, which is due to a further loss of SO_2.

The velocities of decomposition of $MnSO_4$ at 770° and 840° are shown in Fig. 12.

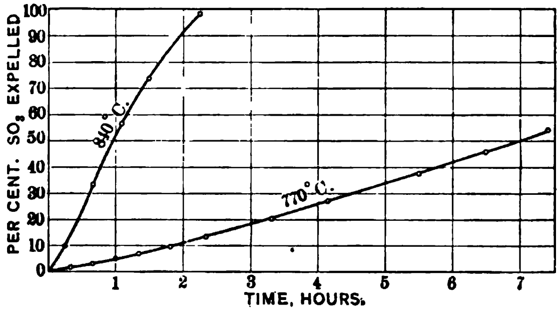

FIG. 12.—VELOCITY OF DECOMPOSITION OF $MnSO_4$ IN A CURRENT OF AIR AT 770° AND 840° C.

There exist three hydrous sulphates: $MnSO_4 + 5H_2O$, $MnSO_4 + 2H_2O$, $MnSO_4 + 1H_2O$. The salt $MnSO_4 + 5H_2O$ drops 3 molecules of H_2O at 25°, the 4th at 60°, and the 5th at 152°. Dissociation begins at 699° and becomes decided at 790°.

7. *Zinc Sulphate*, $ZnSO_4 + 7H_2O$.

Zinc vitriol forms orthorhombic white lustrous crystals, which disintegrate upon exposure to air for any length of time and give off some water.[62] Besides $ZnSO_4 + 7H_2O$, Müller-Erzbach[63] distinguishes the following sulphates: $ZnSO_4 + 2H_2O$, $ZnSO_4 + 1H_2O$, and $ZnSO_4$. Upon heating, zinc vitriol melts in its combined water. According to Wiedemann,[64] heating $ZnSO_4 + 7H_2O$

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[62] Péchard, *Annales de Chimie et de Physique*, Sixth Series, vol. xxviii., p. 415 (1893).

[63] *Annalen der Physik und Chemie*, vol. xxvi., p. 409 (1885).

[64] *Annalen der Physik und Chemie*, vol. xvii., p. 561 (1882).

to 69° converts it into $ZnSO_4 + 6H_2O$; Graham[65] dehydrated it completely at 204°; Hofman[66] found 263° the lowest temperature at which all the water was expelled excepting 0.24 per cent., which could be driven off only at 528°. With a 2-g. sample it took from 30 to 40 hr. to complete a test. Mostowitsch[67] dehydrated the salt at 400° in from 1 to 2 hr. Friedrich[68] found that all the water was expelled at 300°. Heating the anhydrous salt in a current of dry air, Hofman[69] found that with the elimination of the last of the water at 528°, some SO_2 was set free for a short time and then stopped; raising the temperature caused more SO_2 to be liberated in different stages unto complete dissociation, the lowest temperature for which he found to be 739°. It takes a long time to accomplish this result. Mostowitsch[70] decomposed completely 0.6104 g. $ZnSO_4$ at 750° in 27 hr.; it took him only 2 hr. to do the same at 850°. Doeltz-Graumann[71] noted at 700° the first decomposition of $ZnSO_4$. Landis[72] noticed the first traces of SO_2 at 730°; the decomposition was rapid at 760° and apparently complete, as no SO_2 was detected afterward even when the temperature was raised to 980°. Friedrich[73] found his first retardation at 740°, which was caused by a transformation; the second was noticed at 840°, and was due to the formation of the basic salt $3ZnO.2SO_3$; the third, at 935°, was accompanied by the dissociation of the basic salt into ZnO and SO_3. Mostowitsch[74] could not detect a trace of either SO_2 or SO_3 up to 560°; above 560° the test-tube showed the presence of sulphur, and the substance began to lose in weight; the loss at 600° was only 0.40 per cent. of the SO_3 present. The decomposition of about 0.4 g. of $ZnSO_4$ at temperatures ranging from 650 to 900°, heating for 1-hr. periods, is shown in Fig. 13. The curve shows that at 850° the dissociation of $ZnSO_4$ is completed in 1 hr., which is a practical figure. It brings out another point, and this is that up to 700° the expulsion of SO_3 is quick, and above 700° uniformly slow. The diagrams of Hofman[75] tell a similar story, viz., that the dissociation progresses rapidly until one-third of the SO_3 has been expelled, and then more slowly. Both diagrams indicate the presence of an intermediary basic salt, the existence of which has been shown by Friedrich, and verified in the present tests, but at a much lower temperature.

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[65] *Philosophical Magazine*, Third Series, vol. vi., No. 35, p. 333 (May, 1835).
[66] *Trans.*, xxxv., 817 (1904).
[67] *Metallurgie*, vol. viii., No. 24, p. 766 (Dec. 22, 1911).
[68] *Metallurgie*, vol. vii., No. 11, p. 328 (June 8, 1910).
[69] *Loc. cit.*
[70] *Metallurgie*, vol. viii., No. 24, p. 763 (Dec. 22, 1911).
[71] *Metallurgie*, vol. iii., No 13, p. 445 (July 8, 1906).
[72] *Metallurgical and Chemical Engineering*, vol. viii., No. 1, p. 22 (Jan., 1910).
[73] *Metallurgie*, vol. vii., No. 11, p. 328 (June 8, 1910).
[74] *Metallurgie*, vol. viii., No. 24, p. 763 (Dec. 22, 1911).

The result of the chemical analysis of the salt used in the experiments is given in Table VIII.

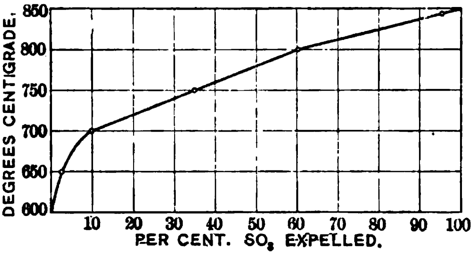

FIG. 13.—DEGREE OF DECOMPOSITION OF $ZnSO_4$ IN 1-HR. PERIODS.

TABLE VIII.—*Composition of* $ZnSO_4 + 7H_2O$.

Component.	Chemical Analysis.	Calculated Analysis.
	Per Cent.	Per Cent.
SO_3	27.90	27.84
ZnO	28.39	28.32
H_2O	43.71 [a]	43.84
SO_3 : ZnO	0.9821	0.9830

[a] By difference.

The zinc was precipitated with sodium carbonate and weighed as ZnO; the S was separated with barium chloride from a chloride solution and weighed as $BaSO_4$.

For the experiment 0.3 g. substance was taken; boat and substance weighed 4.0912 g. It was heated in the thermostat up to 25° for 2.5 hr., whereby the weight was reduced to 4.0725 g.; the loss of 0.0187 g. corresponds to 1 molecule of

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[75] *Trans.*, xxxv., 822 (1904).

H_2O, as indicated in the bottom curve of Fig. 14*a*. Between 28° and 110° a retardation is seen to start at 28° and to extend to 100°; from 40° upward to 69° the curve for the charged boat runs nearly parallel with that for the empty boat, and recedes again suddenly. The charged boat heated to 110° weighed 3.9980 g., showing a loss in weight of 0.0745 g., which corresponds to 4 molecules of H_2O (= 0.0749 g.). Parallel tests with the thermostat showed that heating the original substance to 35° for 24 hr. eliminated 5 molecules of H_2O. Between 110° and 210° a retardation is seen between 115° and 196°. The boat at 210° weighed 3.9800 g.; the loss of 0.0180 g. is equal to the weight of 1 molecule of H_2O (= 0.0187 g.). Heating for an additional 30 min. produced no further change in weight. Between 210° and 501° two absorptions of heat are evident, which extend from 225° to 393°, and from 472° to 501°. The first is due to an almost complete expulsion of the 7th molecule of H_2O, the second to a transformation accompanied

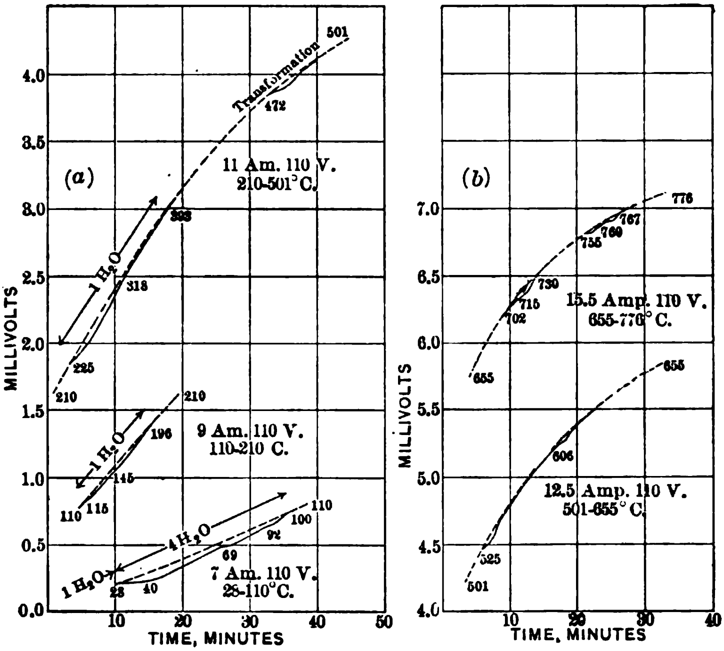

Fig. 14*a*. Fig. 14*b*.

FIGS. 14*a* AND 14*b*.—DEHYDRATION AND DESULPHATIZATION OF $ZnSO_4 + 7H_2O$.

by a slight dissociation, proved by the momentary bluing of the test-solution, and an expulsion of some water. The weight of the boat at 501° was 3.9625 g.; the total loss in weight of 0.1287 g. shows that all the water (= 0.1315 g.) has not yet been driven off. The substance was white, but with a grayish tinge. Between 501° and 655°, Fig. 14*b*, the first retardation, at 525°, is accompanied by a decided emission of SO_2 and an expulsion of the last of the H_2O; the decomposition progresses very slowly up to 606°, when the advance is more quick, but on the whole still very slow. At 655° the boat weighed 3.9605 g. In the last curve, from 655° to 776°, there is noticed an absorption of heat between 702° and 730° accompanied by the splitting-off of both SO_2 and SO_3, which, as shown below, is equal to one-third of the SO_3 present, and corresponds therefore to the formation of the basic salt $3ZnO.2SO_3$. The second thermal effect occurs at 755°, and is due to the decomposition of $3ZnO.2SO_3$. The boat at 776° weighed 3.9536 g. In order to test the formation of this basic salt, 0.3 g. of original substance, boat and salt weighing 3.9776 g., was dehydrated and heated to 730°, the termination of the first retardation, and held there for 4 hr. The salt free from H_2O ought to weigh $3.9776 - 0.1315 = 3.8461$ g.; the weight after 4 hr. at 730° was 3.8186 g.; the difference of 0.0275 g. corresponds to the theoretical value 0.0280 g. for SO_3, if the basic salt $3ZnO.2SO_3$ has been formed, hence we are justified in accepting the formula.

There exist four hydrous salts: $ZnSO_4 + 7H_2O$; $ZnSO_4 + 6H_2O$; $ZnSO_4 + 2H_2O$; $ZnSO_4 + 1H_2O$. The salt $ZnSO_4 + 7H_2O$, heated to 25°, gives up 1 molecule of H_2O, at 28° there are split off 4 molecules, at 115° the 6th, and at 225° the 7th molecule. Heating the anhydrous salt to 702° causes decomposition with the formation of $3ZnO.2SO_3$, which in its turn is dissociated at 755°.

8. *Nickel Sulphate*, $NiSO_4 + 7H_2O$.

Morenosite occurs in the form of apple-green thin prisms; the artificial nickel vitriol forms emerald-green crystals, which weather on exposure to air, losing 1 molecule of H_2O (Dobroserdoff).[76] According to Müller-Erzbach,[77] there exist three hydrous sulphates: $NiSO_4 + 7H_2O$; $NiSO_4 + 6H_2O$; $NiSO_4 + 1H_2O$. Hauer[78] states that heating $NiSO_4 + 7H_2O$ to 100° causes 4 molecules of H_2O to be expelled; Graham[79] found that at 103° there were driven off 6 molecules of H_2O, and that the 7th followed at 279°, while Pierre[80] places the complete dehydration at 200° to 250°. Friedrich[81] found that at 500° the vitriol was completely freed from water.

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[76] Moissan, *Traité de Chimie Minérale*, vol. iv., p. 279 (1905).

[77] *Annalen der Physik und Chemie*, vol. xxvi., p. 409 (1885).

In heating the anhydrous normal salt, a basic salt is said to be formed.[82] Warlimont[83] found that the dissociation of $NiSO_4$ began at 600°, heating for 3 hr. at 625° caused 8.4 per cent. of the SO_4 to be expelled, an additional 3 hr. at 650° gave 33 per cent., and 3 hr. more at 670° gave 88 per cent. Friedrich[84] gives 840° as the beginning of the dissociation.

The nickel vitriol used in the experiments gave, upon chemical analysis, the results recorded in Table IX.

TABLE IX.—*Composition of* $NiSO_4 + 7H_2O$.

Component.	Chemical Analysis.	Calculated Analysis.
	Per Cent.	Per Cent.
SO_3	29.96	29.98
NiO	28.00	28.01
H_2O	42.04[a]	42.01
$SO_3 : NiO$	1.0700	1.0703

[a] By difference.

The nickel was determined electrolytically in an ammoniacal solution; the sulphur was precipated with barium chloride from a chloride solution, and weighed as $BaSO_4$.

For the experiment there was taken 1.0774 g.; the charged boat weighed 8.2456 g. Heating to 32° caused no change in weight. Between 32° and 60°, Fig. 15, retardations were observed at 40° and 52°; the boat heated to 60° weighed 8.2350 g., showing a loss of some water; holding it for 7 hr. at 60° reduced the weight to 8.0850 g., which would correspond to about 2.5 molecules of H_2O. Between 60° and 93° there is seen a retardation at 71°; the boat at 93° weighed 8.0818 g., the salt had become a pale sky-blue; holding it for 135 min. at 93°

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[78] *Sitzungsberichte der Akademie d. Wissenschaften, Wien*, vol. xxxix., p. 305 (1860).

[79] Gmelin-Kraut, *Handbuch der anorganischen Chemie*, vol. v., part ii., p. 72 (1902).

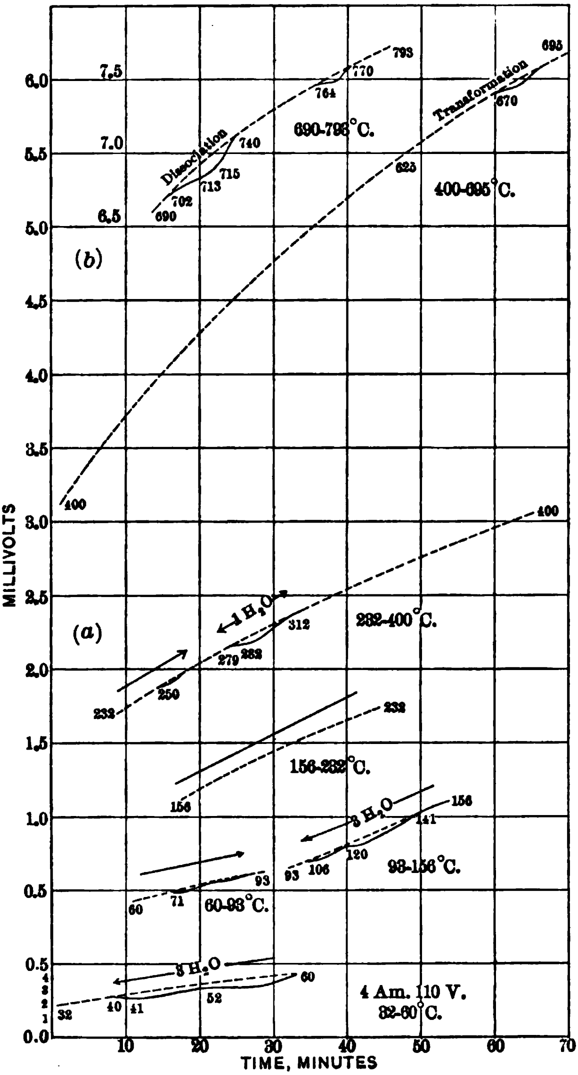
[80] *Annales de Chimie et de Physique*, Third Series, vol. xvi., p. 252 (1846).

[81] *Metallurgie*, vol. vii., No. 11, p. 327 (June 8, 1910).

[82] Tupputi, in Dammer, *Handbuch der anorganischen Chemie*, vol. iii., p. 507 (1893).

[83] *Metallurgie*, vol. vi., No. 4, p. 132 (Feb. 22, 1909).

[84] *Metallurgie*, vol. vii., No. 11, p. 327 (June 8, 1910).



FIGS. 15*a* AND 15*b*.—DEHYDRATION AND DESULPHATIZATION OF $NiSO_4 + 7H_2O$.

reduced the weight to 8.0790 g., showing that 3 molecules of H_2O had been set free; the color begins to change to orange-yellow. Parallel tests in the thermostat showed that heating to 40° needed as much as 110 hr. to expel these 3 molecules of H_2O. In the temperature-range between 93° and 156°, water begins to be removed at 106°, the reaction weakens and becomes again strong at 120°. Heated to 156° the boat weighed 8.0732 g.; holding it for 4.25 hr. at 130° reduced this weight to 7.8740 g., which would correspond to a total expulsion of a little over 5½ molecules of H_2O, or, the second 3 molecules of H_2O had not been entirely set free. Between 156° and 232° the curve of the charged boat about coincides with that of the empty boat. In heating from 232° to 400°, there occurs an absorption of heat at 250°, explained by the driving-off of the rest of the 6th molecule of H_2O. The retardation at 279° is caused by the almost entire removal of the 7th molecule of H_2O; the color of the substance changes to an orange-yellow. The boat brought to 400° and weighing 7.800 g. showed that 0.641 per cent. of 1 molcule of H_2O was still held by the substance. Between 400° and 695° a decided retardation takes place at 670°, unaccompanied by any change in weight; it is due to a transformation which changes the orange-colored salt to a grayish white. In the temperature-range between 690° and 793° there is a thermal effect at 702°, which reaches a maximum at 715°, disappears at 740°, has a duration of 8 min., and shows the dissociation of the salt. With the dissociation takes place the expulsion of the remaining small amount of the 7th molecule of H_2O. In order to verify the dissociation-temperature, a new sample of 0.3 g. of vitriol was taken (weight of charged boat, 5.0082 g.) and heated; at 690° the first SO_3 was set free, but the loss in weight amounted only to 0.1264 g. (7 molecules of $H_2O = 0.1261$ g.); at 740° it rose to 0.1392 g. While the dissociation started at 690°, it became decided at 702°. Between 740° and 793° there was noticed the last retardation, extending from 764° to 770°, which shows an acceleration of the dissociation.

The velocity of decomposition of $NiSO_4$ at 710° and 770° is shown in Fig. 16.

There exist three hydrous salts: $NiSO_4+7H_2O$; $NiSO_4+4H_2O$; $NiSO_4 + 1H_2O$. The salt $NiSO_4 + 7H_2O$ begins to lose 3 molecules of H_2O at 40°, but parts with them very slowly; the same is the case with the second 3 molecules of H_2O, which begin to be set free at 106°; the expulsion of the 7th molecule of H_2O begins at 279°, but is not completed until the dissociation of the salt into NiO and SO_3 begins, which takes place at 702°.

9. *Cobalt Sulphate*, $CoSO_4 + 7H_2O$.

The mineral bieberite forms flesh- to rose-red monoclinic crystals; those of the prepared cobalt vitriol are similar. They are stable if separated from a neutral solution, unstable if the solution was acid.[85] Müller-Erzbach[86] distinguishes three hydrates: $CoSO_4 + 7H_2O$; $CoSO_4 + 6H_2O$; $CoSO_4 + 1H_2O$. Friedrich[87] found that all the water was expelled at 420°, and that dissociation began at 880°. Hartley[88] says that the anhydrous salt is unchanged at 300°, becomes lilac-colored between 500° and a red-heat, but regains its original rose-color upon cooling. According to Barth,[89] the anhydrous salt is stable at 600°, heating for 3 hr. at 613° caused a decomposition of 3.4 per cent. of the salt, and for 2 hr. at 735° the complete dissociation.

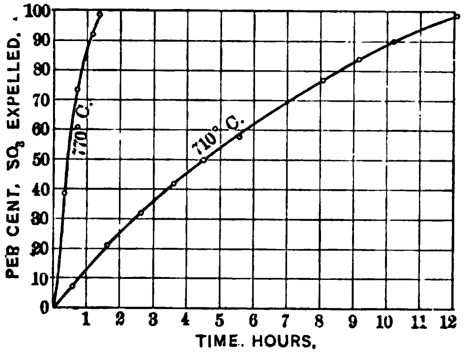


FIG. 16.—VELOCITY OF DECOMPOSITION IN A CURRENT OF AIR OF $NiSO_4$ AT 710° AND 770° C.

A chemical analysis of the salt used in the experiments gave the results shown in Table X.

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[85] Baubigny and Péchard, *Comptes rendus de l'Académie des Sciences*, vol. cxv., No. 3, p. 171 (1892).

[86] *Annalen der Physik und Chemie*, vol. xxvi., p. 409 (1885).

[87] *Metallurgie*, vol. vii., No. 11, p. 326 (June 8, 1910).

[88] *Journal of the Chemical Society*, vol. lxxxiii., p. 404 (1903).

[89] *Metallurgie*, vol. ix., No. 6, p. 202 (Mar. 22, 1912).

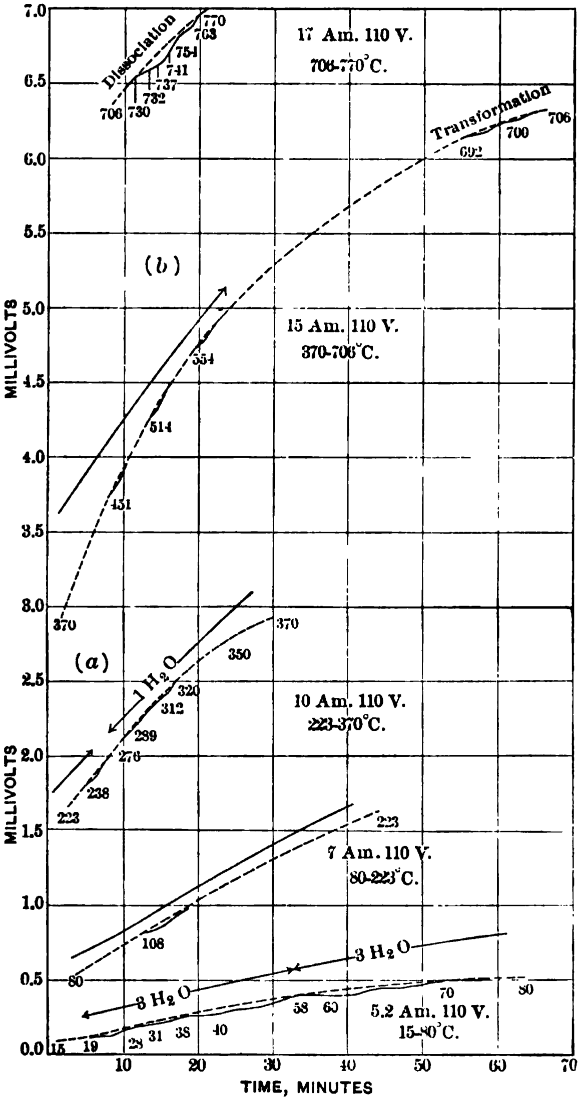
TABLE X.—*Composition of* $CoSO_4 + 7H_2O$.

Component.	Chemical Analysis.	Calculated Analysis.
	Per Cent.	Per Cent.
SO_3	28.54	28.53
CoO	26.57	26.59
H_2O	44.89 [a]	44.88
SO_3 : CoO	1.0741	1.0728

[a] By difference.

The cobalt was determined electrolytically from an ammoniacal solution; the sulphur was precipitated with barium chloride from a chloride solution and weighed as $BaSO_4$.

The weight of the substance used for experiment was 0.3995 g.; the charged boat weighed 6.7860 g. The bottom curve, Fig. 17, representing the heating from 15° to 80°, shows two leading retardations, at 19° and at 58°. At 19° water begins to be separated, the expulsion becomes weaker at 28° and 38°, gains in force up to 40°, diminishes, and finally stops at 58°. Parallel tests in the thermostat showed that heating for 30 hr. at 36° caused the separation of 3 molecules of H_2O. At 58° begins the splitting-off of an additional 3 molecules of H_2O. Heated to 70° the charged boat weighed 6.6748 g., showing a loss of 0.1112 g. of H_2O, which would correspond to a total of 4⅓ molecules. Heating at 80° for 1 hr. increased the loss in weight; the boat weighed 6.6450 g., showing a decrease of 0.0298 g.; continuing the heating at 80° for 45 min. reduced the weight only very little, viz., 6.6444 g. The salt is homogeneous and purple-colored. In the temperature-range between 80° and 223° there is a thermic effect accompanied by a slight further reduction in weight, the boat, heated at 223°, weighing 6.6388 g.; holding it at 223° for an additional 30 min. produced no further expulsion of water. Between 223° and 370° the first absorption of heat is seen at 238°, which is due to the removal of the remaining water of the 6th molecule. At 276° the last molecule begins to be set free and is apparently completely liberated at 320°, when the boat weighed 6.6106 gr., showing that there had separated altogether 0.1754 g., which is less than 0.1793 g., the weight of 7 molecules of H_2O. Heating to 340° caused the weight of the boat to fall to 6.6082 g., still not enough to form 1 molecule of H_2O. The salt was blue when hot, purple to rose-color when cold.



FIGS. 17*a* AND 17*b*.—DEHYDRATION AND DESULPHATIZATION OF $CoSO_4 + 7H_2O$.

Raising the temperature from 370° to 706°, Fig. 17, causes three minor retardations, at 451°, 514° and 554°, which are caused by the removal of the rest of the 7th molecule of H_2O.

The weight of the boat at 706° was 6.6072 g., showing that 0.1788 g. had been expelled; the theoretical value is 0.1792 g. The heat-absorption seen at 692° is due to a transformation, as there is no change in weight nor any reaction with the indicator. In the last curve, covering the range from 706° to 770°, there is a strong thermal effect at 732°, accompanied by the setting-free of SO_2. The bluing of the indicator stopped at 737°, but started again at 754°. At 770° the boat weighed 6.6054 g., equal to a loss in weight of 0.1806 g. As the 7 molecules of H_2O of the substance weigh 0.1792 g., the difference between the two numbers shows that SO_2 has been expelled. In order to verify the temperature of expulsion of the last molecule of H_2O, and of the dissociation of the anhydrous salt, 0.4208 g. of fresh vitriol was taken; the charged boat weighed 6.8072 g. Heating for 1 hr. at 650° caused the weight to be reduced to 6.7384 g.; the difference, 0.1683 g., corresponds to the theoretical weight of 7 molecules of H_2O, 0.1689 g., showing that at 650° the dehydration was complete. Heating to 690° and 700° for 1 hr. caused no change in weight nor any effect on the indicator; between 717° and 720° the indicator becomes blue after 15 min.; after holding 1 additional hour at this temperature, the loss in weight was 0.0030 gr.; 1 hr. at 725° caused a loss of 0.0128 g. It is therefore correct to say that the dissociation begins at 720°.

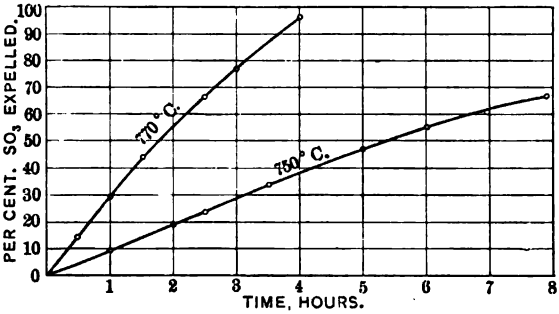


FIG. 18.—VELOCITY OF DECOMPOSITION OF $CoSO_4$ IN A CURRENT OF AIR AT 750° AND 770° C.

The velocity of decomposition at 750° and 770° is shown in Fig. 18.

There exist three hydrous salts: $CoSO_4 + 7H_2O$; $CoSO_4 +$

$4H_2O$; $CoSO_4 + 1H_2O$. The salt $CoSO_4 + 7H_2O$ begins to lose 3 molecules of H_2O at 19°, 3 more are expelled at 58°, and the last one is set free at 276°. The anhydrous salt undergoes a transformation at 692°, and is dissociated at 720°.

10. *Cadmium Sulphate*, $3CdSO_4 + 8H_2O$.

The formula of Richter[90] for hydrous cadmium sulphate crystallizing out at ordinary temperature has been used in the heading. The crystals readily disintegrate when exposed to air. Abegg[91] doubts the existence of any other hydrous salt, excepting $CdSO_4 + 1H_2O$; the existence of this has been proved by Lescœur[92]. According to Frémy,[93] the salt is dehydrated at a dark red. Mendelejeff[94] states that $CdSO_4$ is less readily decomposed by heat than $ZnSO_4$.

A chemical analysis of the substance used in the experiments gave the results shown in Table XI.

TABLE XI.—*Composition of* $CdSO_4 + 8H_2O$.

Component.	Chemical Analysis.	Calculated Analysis.
	Per Cent.	Per Cent.
SO_3	31.11	31.17
CdO	50.20	50.11
H_2O	18.69 [a]	18.72
$SO_3 : CdO$	0.6197	0.6223

[a] By difference.

The cadmium was determined electrolytically from a potassium cyanide solution according to the method of Beilstein and Jawein; the sulphur was precipitated from a chloride solution with barium chloride and weighed as $BaSO_4$.

For the experiment 0.3 g. of substance was used; the charged boat weighed 5.0078 g. The bottom curve, Fig. 19*a*, covering the temperature-range from 30° to 110°, shows that water was split off already at 30°, and that the thermic effect ceased at 39°; the second separation of water began at 41°, and lasted until 90° was reached. Heated to 110° the boat weighed 4.9730° g.; further heating for 1 hr. did not cause any change in weight. The loss in water at 110° was, therefore, 0.0348 g., which corresponds to an expulsion of $1\frac{3}{8}$ molecules of H_2O. Parallel tests in the thermostat for 14 hr. at 37° showed that $\frac{3}{8}$ molecule of H_2O had been driven off, and for 15 hr. at 60° that 1 molecule of H_2O had been separated. In the range between 110° and 260° a retardation occurs at 170°, which extends to 220°. As the charged boat heated to 220° weighed 4.9512 g., the loss in weight corresponded to 1 molecule of H_2O.

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[90] *Lehrbuch der anorganischen Chemie*, bearbeitet von Klinger, Bonn, p. 386 (1906).

[91] *Handbuch der anorganischen Chemie*, vol. ii., part 2, p. 469 (1905.)

[92] *Annales de Chimie et de Physique*, Seventh Series, vol. iv., p. 222 (1895).

[93] *Encyclopédie Chimique*, vol. iii., p. 285 (1900).

[94] *Grundlagen der Chemie*, p. 714 (1891).

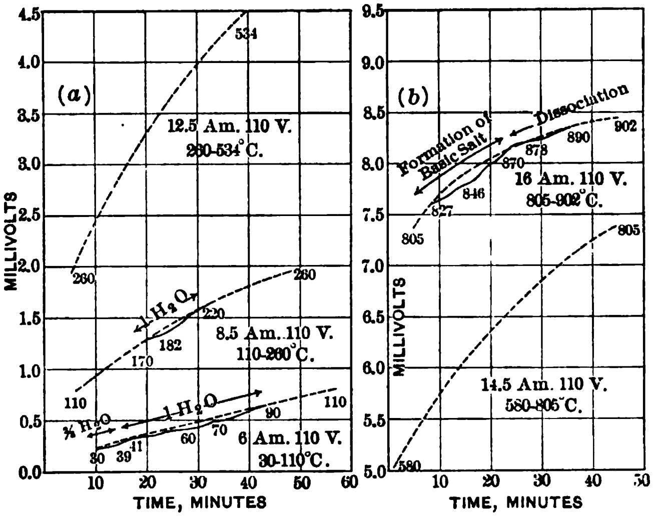

Fig. 19*a*. Fig. 19*b*.

FIGS. 19*a* AND 19*b*.—DEHYDRATION AND DESULPHATIZATION OF $CdSO_4 + 8H_2O$.

Between 260° and 580° (in Fig. 19*a* the curve is drawn only to 534°), and 580° and 805°, Fig. 19*b*, no thermal effects were noticed, but between 805° and 902° are seen two retardations; the first, lasting from 827° to 870°, is due to the formation of a basic salt; the second, extending from 878° to 890°, is caused by the dissociation of the basic salt into CdO and SO_3. The boat heated to 902° weighed 4.9460 g.; the loss in weight was 0.058 g. Heating at 902° for 9 hr. reduced the weight to 4.8600 g.; there had been expelled 0.0912 g.; the theoretical value for SO_3 is 0.0935 g. The salt was white when the prolonged heating at 902° began, became yellow when hot, but turned again white upon cooling. After heating 4 hr. black specks were noticed in the white salt; these increased until at the end of the 9th hr. the substance became black. In order to verify the formation of the basic salt, another experiment with 0.3 g. of substance was made. The charged boat, weighing 3.6150 g., was heated to 850°, and held there for 12 hr.; its weight was reduced to 3.4840 g., which corresponds, after deducting the 0.0561 g. of H_2O expelled, to a loss of 0.0749 g. of SO_3, that is, four-fifths of the total SO_3. The formula for the basic salt is therefore $5CdO.SO_3$. Heating for two additional hours did not change the weight of the white salt. Raising the temperature to 880°, that is, above the starting-point of the second retardation in the curve, causes expulsion of SO_3, and a change of the white color of the basic salt into the black of the oxide. The velocity of decomposition of $3CdSO_4$ at 960° is shown in Fig. 20.

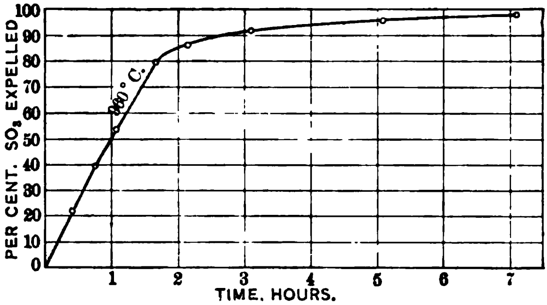

FIG. 20.—VELOCITY OF DECOMPOSITION IN A CURRENT OF DRY AIR OF $CdSO_4$ AT 960° C.

There exist three hydrous salts: $3CdSO_4 + 8H_2O$; $CdSO_4 + 2H_2O$; $CdSO_4 + 1H_2O$. Heating $3CdSO_4 + 8H_2O$ to 30° causes two-thirds of the H_2O to be expelled, at 41° 1 molecule of H_2O separates off, and at 170° another. The anhydrous sulphate is decomposed at 827° to form the basic salt $5CdO.SO_3$, and this is finally dissociated into CdO and SO_3 at 878°.

11. *Magnesium Sulphate, $MgSO_4 + 7H_2O$.*

The natural as well as the artificial epsom salt forms white, translucent, orthorhombic crystals, which weather in dry, but remain unaltered in moist, air (Abegg).[95] Warming causes them to melt in their water of crystallization. Heating to 50° gives a transition form of $MgSO_4 + 6H_2O$ (Van der Heide).[96] According to Graham,[97] heating to 100° furnishes crystals of $MgSO_4$ with a little more than 2 molecules of H_2O; at the same temperature in vacuo, in the presence of H_2SO_4, the slight excess of H_2O above 2 molecules is completely removed. He found that the 6th molecule of H_2O was expelled at 132°, and that the 7th was retained even at 237°. Bailey[98] says that the anhydrous salt is stable at 450°; Le Chatelier[99] determined its dissociation-temperature as 1,160°.

The composition of the salt used in the experiments is given in Table XII.

TABLE XII.—*Composition of* $MgSO_4 + 7H_2O$.

Component.	Chemical Analysis.	Calculated Analysis.
	Per Cent.	Per Cent.
SO_3	33.30	33.70
MgO	16.76	16.72
H_2O	49.94[a]	50.08
SO_3 : MgO	1.9318	1.9258

[a] By difference.

The magnesia was precipitated in the cold from an ammoniacal solution with sodium phosphate, ignited, and weighed as $Mg_2P_2O_7$; the sulphur was precipitated from a chloride solution with barium chloride and weighed as $BaSO_4$.

For the experiment there was taken 0.3 g.; the charged boat weighed 3.9773 g. In the bottom curve, Fig. 21*a*, extending from 17° to 142°, the first retardation occurs at 19°; it is slight, but is caused by the splitting-off of 1 molecule of H_2O. A parallel test in the thermostat showed that heating 0.3 g. of substance at 19° for 8.5 hr. expelled 1 molecule of H_2O. The second retardation, extending from 38° to 105°, is much more decided than the first; it corresponds to the driving-off of 4 molecules of H_2O. Heating 0.3 g. in the thermostat at 38° for 43.5 hr. removed a total of 5 molecules of H_2O. At 112° is a third retardation, caused by the loss of 1 molecule of H_2O. The charged boat heated to 142° weighed 3.8638 g.; holding it for 30, 25, and 30 min. at this temperature decreased the weights to 3.8486, 3.8483, and 3.8483 g., showing that 0.1290 g. had been lost; the theoretical figure for loss of H_2O is 0.1284 g. Heating in the thermostat for 3 hr. at 120° expelled 6 molecules of H_2O. In the second curve, extending from 142° to 445°, there occurs an absorption of heat at 203°, which lasts until the temperature has risen to 302°. The charged boat heated to 445° weighed 3.8260 g., corresponding to a total loss of 0.1509 g.; the weight of 7 molecules of H_2O in 0.3 g. of substance is 0.1509 g.; the salt, therefore, is completely dehydrated; it is white, with slight brownish shade. Raising the temperature from 445° to 639° and from 639° to 834° shows no thermal effects in the curves of Fig. 21*b*, the salt remains white. The desulphatization of the anhydrous salt is shown in Fig. 21*c*. The curve extends from 834° to 1,000°. At 890° there is seen a small deflection of the curve of the charged from that

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[95] *Handbuch der anorganischen Chemie*, vol. ii., part 2, p. 58 (1905).

[96] *Zeitschrift für physikalische Chemie*, vol. xii., p. 417 (1893).

[97] *Philosophical Magazine*, Third Series, vol. vi., No. 36, p. 421 (June, 1835).

[98] *Journal of the Chemical Society*, vol. li., p. 676 (1887).

[99] *Bulletin de la Société Chimique*, vol. xlvii., p. 300 (1887).

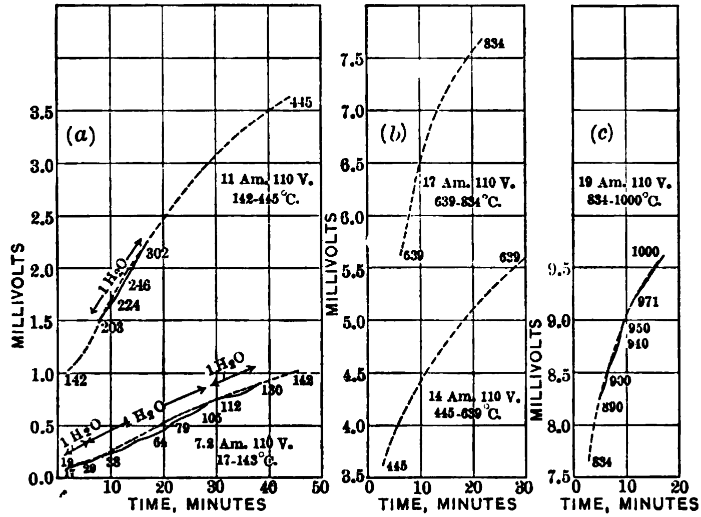

Fig. 21*a*. Fig. 21*b*. Fig. 21*c*.

FIGS. 21*a*, 21*b*, AND 21*c*.—DEHYDRATION AND DESULPHATIZATION OF $MgSO_4 + 7H_2O$.

of the empty boat; it increases from 900° to 940°, decreases, and gains again in magnitude at 950°. The indicator becomes colored at 900°. The dissociation begins at 890° and becomes decided at 971°.

Fig. 22 shows the velocity of decomposition of the anhydrous salt at 1,000°.

There exist four hydrous salts: $MgSO_4 + 7H_2O$; $MgSO_4 + 6H_2O$; $MgSO_4 + 2H_2O$; $MgSO_4 + 1H_2O$. Upon heating $MgSO_4 + 7H_2O$ to 19° there is set free 1 molecule of H_2O, at 38° 4 molecules are liberated, the 6th molecule is split off at 112°, and the salt is dehydrated at 203°. Heating the anhydrous salt to 890° causes dissociation to begin.

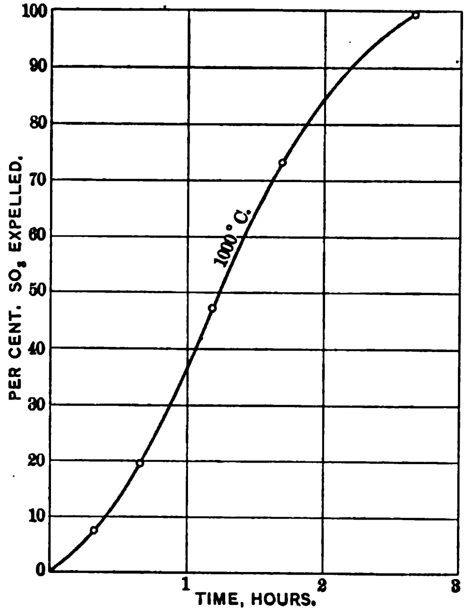

FIG. 22.—VELOCITY OF DECOMPOSITION IN A CURRENT OF AIR OF $MgSO_4$ AT 1,000° C.

12. *Silver Sulphate, Ag_2SO_4.*

This salt forms small white lustrous crystals. It is known to fuse when heated and to be decomposed at a temperature that lies higher than its melting-point. The only determinations of these temperatures are those by Bradford [100] and Friedrich.[101] Bradford determined the freezing-point at 655° and the dissociation at 1,095°, although he noticed losses in weight already at 900°, which increased with the temperature and the duration of the heating. Friedrich observed a transformation at 410°, when the white salt turned yellow; determined the melting-point at 660° and the dissociation-temperature at 1,085°.

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[100] *Trans.*, xxxiii., 50 (1902).

A chemical analysis of the salt used in the experiments gave the results shown in Table XIII.

TABLE XIII.—*Composition of* Ag_2SO_4.

Component.	Chemical Analysis.	Calculated Analysis.
	Per Cent.	Per Cent.
SO_3	25.71	25.67
Ag_2O	74.21	74.33
$SO_3 : Ag_2O$	0.3464	0.3453

The silver was precipitated with dilute hydrochloric acid and weighed as AgCl, the sulphur with barium chloride from a chloride solution and weighed as $BaSO_4$.

In the experiments 0.3 g. of substance was used; the charged boat weighed 4.0924 g. Heating for 1.5 hr. to 260° gave a loss in weight of 0.0007 g.; that is, a trace of moisture was expelled. The lower curve, Fig. 23*a*, covers the range of temperature from 268° to 445°; at 417° is a decided retardation, caused by a transformation, as the weight remained unchanged (4.0917 g.) and the indicator was unaffected, but the color of the salt changed from white to a grayish yellow. In the upper curve, Fig. 23*a*, a thermal effect is noticed at 656°; the weight has not changed (4.0917 g.), but the salt has fused. Between 690° and 820°, Fig. 23*b*, there is only a slight thermal effect, which begins at 752° and extends to 917°; the weight is reduced very little, and the indicator shows that SO_2 is liberated in very small amounts. At 917° there is a strong absorption of heat; the full-drawn line runs horizontally for 3 min. and then bends upward; the indicator shows quickly the presence of much SO_2. When the first evolution of gas has ceased, it will appear again when the temperature has risen and last for about 4 min.; it continues to do this until the dissociation is complete. The weight of the charged boat decreased at 917° from 4.0917 to 4.0778 g.; heating for three 30-min. periods at 940° reduced it to 4.0466, 4.0188, and 3.9910 g., when the salt had been complely converted into metallic silver.

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[101] *Metallurgie*, vol. vii., No. 11, p. 330 (June 8, 1910).

The results show that a transformation takes place at 417°, fusion at 656°, and dissociation into Ag_2 and $SO_2 + O$ begins at 752° and becomes decided at 917°.

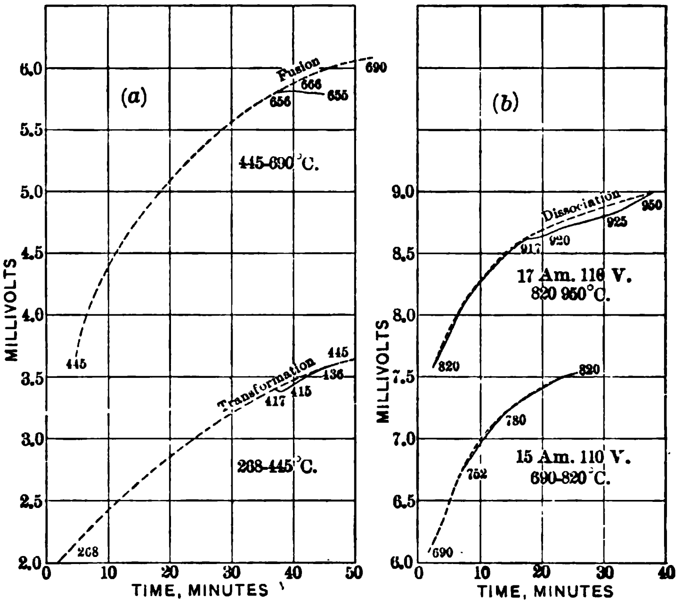

Fig. 23a. Fig. 23b.

FIGS. 23a AND 23b.—DESULPHATIZATION OF Ag_2SO_4.

13. *Calcium Sulphate*, $CaSO_4 + 2H_2O$.

The mineral gypsum forms monoclinic white crystals with a pearly to shiny luster, the prepared salt a white voluminous crystalline precipitate. According to Kraut and Precht,[102] $CaSO_4 + 2H_2O$ loses its water at 100°, while Millon[103] maintains that it loses 1.5 molecules of H_2O at 110° and is dehydrated at 300°. Le Chatelier[104] found that gypsum was dehydrated at 160°. The research of van't Hoff[105] has shown that gypsum heated in a closed vessel to 107° at a pressure of 970 mm. lost 1.5 molecules of H_2O, or rather that it formed the compound $2CaSO_4 + H_2O$. This is confirmed by Le Chatelier.[106] According to Davis,[107] $CaSO_4 + 2H_2O$ undergoes a transformation upon heating before it gives off any water. Above 600° $CaSO_4$ is dead-burnt; that is, it cannot take up any more water (Rohland).[108] Potilizin[109] found that $CaSO_4$ can recombine with water only if it has not been heated to above 200°.

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[102] Liebig's *Annalen*, vol. clxxviii., p. 129 (1875).

[103] *Annalen der Chemie und Physik*, vol. xix., p. 222 (1847).

[104] *Annales des Mines*, Eighth Series, vol. xi., p. 345 (1887).

The chemical analysis of the substance used in the experiments is given in Table XIV.

TABLE XIV.—*Composition of* $CaSO_4 + 2H_2O$.

Component.	Chemical Analysis.	Calculated Analysis.
	Per Cent.	Per Cent.
SO_3	46.99	47.07
CaO	32.00	32.01
H_2O	21.01 [a]	20.92
SO_3 : CaO	1.4684	1.4704

[a] By difference.

The lime was precipitated with ammonium oxalate, the calcium oxalate ignited and weighed as CaO; the sulphur was precipitated from a chloride solution with barium chloride and weighed as $BaSO_4$.

In the dehydration experiments there was used 0.2 g. of substance; the charged boat weighed 4.2402 g. The bottom curve, Fig. 24, covers the temperature-range from 16° to 110°. No loss in weight was experienced up to 30°. This was confirmed by holding another sample just below 30° for a considerable time. At 30° is seen the first retardation, which extends to 77°. The second thermal effect is seen at 80°; it extends to 110°. The charged boat heated to 110° weighed

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[105] *Zeitschrift für physikalische Chemie*, vol. xlv., p. 257 (1903).

[106] *Comptes rendus de l'Académie des Sciences*, vol. xcvi., No. 23, p. 1668 (1883).

[107] *Journal of the Society of Chemical Industry*, vol. xxvi., No. 13, p. 727 (July I5, 1907).

[108] *Zeitschrift für anorganische Chemie*, vol. xxxvi., No. 3, p. 332 (Sept. 16, 1903).

[109] *Journal der russischen physikalisch-chemischen Gesellschaft*, vol. xxvi., p. 170 (1894).

4.2238 g.; holding for 30 min. at this temperature reduced the weight to 4.2090 g. There had been expelled 0.0312 g. of H_2O, which corresponds to 1.5 molecules of H_2O (= 0.0313 g.). Tests in the thermostat at 33° for 40 hr. showed no loss in water. The retardation at 30° must be ascribed to a transformation followed by an expulsion of water. In the thermostat 1 molecule of H_2O was expelled at 38°; this is indicated in the curve by a very slight retardation. In the upper curve, representing the temperature-range from 110° to 344°, there

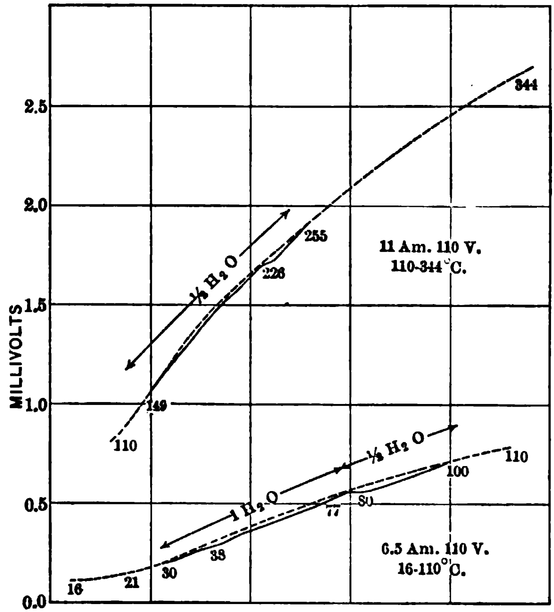

Fig. 24.—Dehydration of $CaSO_4 + 2H_2O$.

is seen a retardation, which begins at 149° and stops at 255°; it represents the expulsion of the remaining 0.5 molecule of H_2O. The boat heated to 344° weighed 4.2056 g.; heating for 1 additional hour at 340° reduced the weight to 4.2054 g. There had been expelled 0.0348 g., which corresponds to 2 molecules of H_2O (= 0.0348 g.).

The dissociation of anhydrous $CaSO_4$ has been studied by Hofman and Mostowitsch,[110] who found that dissociation began at 1,200°; that the salt fused at 1,360°, being more or less decomposed; that the presence of SiO_2 lowered the beginning of the dissociation to 1,000°, and that the formation of $CaSiO_4$ was finished at 1,250°; that Fe_2O_3 acted similarly to SiO_2, the dissociation beginning at 1,100°, and finishing at 1,250°.

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[110] *Trans.*, xxxix., 628 (1908).

14. *Barium Sulphate, BaSO$_4$.*

The mineral barite forms white orthorhombic crystals having a vitreous to resinous luster; the artificial product is a white crystalline substance when precipitated from a hot acid solution.

The dissociation of this salt has been studied by Mostowitsch,[111] who found that the prepared salt gives off SO_3 at 1,510°, fuses at 1,580°, but is not completely decomposed even then, as the fused mass is a mixture of BaO and $BaSO_4$. He ascertained that the presence of a small amount of foreign substance lowered the dissociation-temperature; also that SiO_2 caused decomposition to begin at 1,000°, and that with Fe_2O_3 it started at 1,100°.

V. HYDROUS SULPHATES IN GENERAL.

Our experiments have shown that in subjecting a hydrous sulphate to a gradually increasing temperature all the water of crystallization is split off and carried away by the current of dry air passing through the heating-tube. This separation of water does not take place continuously, but in a series of steps. Thus it has been shown that at a given temperature a sulphate with a certain amount of water of crystallization exists, independently of the other hydrous salts that the sulphate is capable of forming. The hydrous sulphate is a definite chemical compound which takes part as such in chemical reactions. An interesting fact, noted first by Graham[112] and brought out in these experiments, is that, as a rule, the last molecule of H_2O requires for its separation a much higher temperature than do the others, which usually are split off below the boiling-point of water, and which are separated from one another by small intervals of temperature.

The results of our experiments upon dehydration are brought together in Table XV.

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[111] *Metallurgie*, vol. vi., No. 14, p. 450 (July 22, 1909).

[112] *Philosophical Magazine*, Third Series, vol. vi., No. 35, p. 327 (May, 1835).

TABLE XV.—*Dehydration of Metallic Sulphates.*

	Temperature of Beginning of Dehydration.	Product Formed.	Remarks.
	Degrees C.		
$FeSO_4 + 7H_2O$	21	$FeSO_4 + 4H_2O$	Slight apple-green.
$FeSO_4 + 4H_2O$	80	$FeSO_4 + H_2O$	White.
$FeSO_4 + H_2O$	406	$Fe_2O_3 + 2SO_3$	Yellowish brown.
$Al_2(SO_4)_3 + 16H_2O$	51	$Al_2(SO_4)_3 + 13H_2O$	White.
$Al_2(SO_4)_3 + 13H_2O$	82	$Al_2(SO_4)_3 + 10H_2O$	White.
$Al_2(SO_4)_3 + 10H_2O$	97	$Al_2(SO_4)_3 + 7H_2O$	White.
$Al_2(SO_4)_3 + 7H_2O$	109	$Al_2(SO_4)_3 + 4H_2O$	White.
$Al_2(SO_4)_3 + 4H_2O$	180	$Al_2(SO_4)_3 + H_2O$	White.
$Al_2(SO_4)_3 + H_2O$	316	$Al_2(SO_4)_3$	White.
$CuSO_4 + 5H_2O$	27	$CuSO_4 + 3H_2O$	Sky-blue.
$CuSO_4 + 3H_2O$	93	$CuSO_4 + H_2O$	Pale blue.
$CuSO_4 + H_2O$	155	$CuSO_4$	White.
$MnSO_4 + 5H_2O$	25	$MnSO_4 + 2H_2O$	Pale peachblossom.
$MnSO_4 + 2H_2O$	60	$MnSO_4 + H_2O$	Paler than preceding.
$MnSO_4 + H_2O$	152	$MnSO_4$	Paler than preceding.
$ZnSO_4 + 7H_2O$	25	$ZnSO_4 + 6H_2O$	White.
$ZnSO_4 + 6H_2O$	28	$ZnSO_4 + 2H_2O$	White, granular.
$ZnSO_4 + 2H_2O$	115	$ZnSO_4 + H_2O$	White.
$ZnSO_4 + H_2O$	225	$ZnSO_4$	White.
$NiSO_4 + 7H_2O$	40	$NiSO_4 + 4H_2O$	Green.
$NiSO_4 + 4H_2O$	106	$NiSO_4 + H_2O$	Yellow.
$NiSO_4 + H_2O$	279	$NiSO_4$	Orange colored.
$CoSO_4 + 7H_2O$	19	$CoSO_4 + 4H_2O$	Rose.
$CoSO_4 + 4H_2O$	58	$CoSO_4 + H_2O$	Lilac.
$CoSO_4 + H_2O$	276	$CoSO_4$	Lilac.
$CdSO_4 + \frac{8}{3}H_2O$	30	$CdSO_4 + 2H_2O$	White.
$CdSO_4 + 2H_2O$	41	$CdSO_4 + H_2O$	White.
$CdSO_4 + H_2O$	170	$CdSO_4$	White.
$MgSO_4 + 7H_2O$	19	$MgSO_4 + 6H_2O$	White.
$MgSO_4 + 6H_2O$	38	$MgSO_4 + 2H_2O$	White.
$MgSO_4 + 2H_2O$	112	$MgSO_4 + H_2O$	White.
$MgSO_4 + H_2O$	203	$MgSO_4$	White.
$CaSO_4 + 2H_2O$	30	$CaSO_4 + H_2O$	White.
$2CaSO_4 + 2H_2O$	80	$2CaSO_4 + H_2O$	White.
$2CaSO_4 + H_2O$	149	$2CaSO_4$	

In examining the several hydrous sulphates from the point of view of the periodic system (Mendelejeff table, p. 943), it is seen that the sulphates of each group have certain characteristics. Using the notation of Müller-Erzbach[113] to represent the splitting-off of the several molecules of water, there appears the following:

For group VIII., to which belong the metals Fe, Co, Ni, Cu, we have the formula:

$$\frac{MSO_4 + 1H_2O}{+ nH_2O} + nH_2O, \text{ with } n = 3 \text{ for Fe,Co,Ni, and } n = 2 \text{ for Cu.}$$

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[113] *Annalen der Chemie und Physik*, vol. xix., p. 135 (1896).

Of the 7 molecules of water of $CoSO_4 + 7H_2O$, for example, there are first expelled 3 molecules of H_2O, then 3 more, and finally 1 molecule.

For group VII., to which belongs Mn, we have the formula:

$$\frac{MSO_4 + 1H_2O}{+ 1H_2O} + nH_2O, \text{ with } n = 3$$

Of the 5 molecules of water of $MnSO_4 + 5H_2O$, there are first expelled 3 molecules, then 1 molecule, and again 1 molecule.

For group III., to which belongs Al, we have the formula:

$$\cfrac{\cfrac{\cfrac{MSO_4 + 1H_2O}{+ nH_2O} + nH_2O}{+ nH_2O} + nH_2O}{+ nH_2O}, \text{ with } n = 3$$

Of the 16 molecules of water of $Al_2(SO_4)_3 + 16H_2O$, there are expelled 3 molecules in each of the 5 different stages, when the last molecule follows suit.

For group II., to which belong Mg, Ca, Zn, Cd, we have the general formula:

$$\cfrac{\cfrac{MSO_4 + 1H_2O}{+ 1H_2O} + nH_2O}{+ 1H_2O}, \text{ with } n = 4$$

Of the 7 molecules of water of $MgSO_4 + 7H_2O$, there is first expelled 1 molecule, then follow 4, afterward are split off separately the 6th and the 7th.

It is further seen that in each group there is found with the rise of the atomic weight a fall in the water of crystallization. Thus in group VIII., $(Fe, Ni, Co)SO_4$ have 7 molecules of H_2O, while $CuSO_4$ has only 5; in group II. $(Mg, Zn)SO_4$ have 7 molecules, while $CdSO_4$ has only $\frac{8}{3}$ molecules. The metals having the largest atomic weights, Ag, Pb, and Ba, are anhydrous.

In each group the even-numbered series has less water of crystallization than the odd-numbered. Thus in group II, $(Ca, Ba)SO_4$ have less combined water than $(Mg, Zn, Cd)SO_4$.

VI. ANHYDROUS SULPHATES IN GENERAL.

The amounts of heat necessary to decompose the several anhydrous sulphates vary greatly; some are dissociated at low temperatures, others at high. The results of our experiments are assembled in Table XVI., in which they are arranged in the order in which they are dissociated with increasing temperatures. In one column are given the temperatures at which decompositions start, in another the temperatures at which they become energetic; the second column, therefore, gives temperatures which are available for furnace-work. The table shows that some sulphates are directly converted into oxides, while the dissociation of others first leads to a basic salt stable at the temperature at which the first SO_3 is driven off. The basic sulphates obtained in the present series of experiments are: $Fe_2O_3.2SO_3$; $5Bi_2O_3.4(SO_3)_3$; $2CuO.SO_3$; $6PbO.5SO_3$; $3ZnO.2SO_3$; $5CdO.SO_3$.

TABLE XVI.—*Desulphatization of Anhydrous Metallic Sulphates.*

Metallic Sulphates.	Temperature of Beginning of Decomposition.	Temperature of Energetic Decomposition.	Products of Decomposition.	Remarks.
	Degrees C.	Degrees C.		
$FeSO_4$	167	480	$Fe_2O_3.2SO_3$	Yellow-brown.
$Fe_2O_3.2SO_3$	492	560	Fe_2O_3	Red.
$Bi_2(SO_4)_3$	570	639	$5Bi_2O_3.4(SO_3)_3$	White.
$Al_2(SO_4)_3$	590	639	Al_2O_3	White.
$PbSO_4$	637	705	$6PbO.5SO_3$	White.
$CuSO_4$	653	670	$2CuO.SO_3$	Orange-color.
$MnSO_4$	699	790	Mn_3O_4	Dark red to black.
$ZnSO_4$	702	720	$3ZnO.2SO_3$	White, cold and not.
$2CuO.SO_3$	702	736	CuO	Black.
$NiSO_4$	702	764	NiO	Brownish green.
$CoSO_4$	720	770	CoO	Brown to black.
$3ZnO.2SO_3$	755	767	ZnO	Hot yellow, cold white.
$CdSO_4$	827	846	$5CdO.SO_3$	White.
$5Bi_2O_3.4(SO_3)_3$	870	890	Bi_2O_3(?)	Yellow.
$5CdO.SO_3$	878	890	CdO	Black.
$MgSO_4$	890	972	MgO	White.
Ag_2SO_4	917	925	Ag	Silver white.
$6PbO.5SO_3$	952	962	$2PbO_2SO_3$ (?)	White to yellow.
$CaSO_4$	1,200	....	CaO	White.
$BaSO_4$	1,510	....	BaO	White.

Considering the formation of basic salt from the point of view of the periodic system, it appears that in each group the tendency to form basic salts increases with the rise of the atomic weight. Thus in group VIII., in the iron series, Cu, with the largest atomic weight, forms a basic salt, while Fe, with the smallest atomic weight, forms no ferrous basic salt; normal ferric salt also forms no basic salt. In groups V. and

IV., the metals with the highest atomic weights, that is, Bi and Pb, form basic salts. In group II., $MgSO_4$ is dissociated into MgO and SO_3, while (Zn, Cd)SO_4 form basic salts.

Independently of the periodic system, there will be noted further characteristics: Normal sulphates which do not form basic salts upon dissociation show both SO_2 and SO_3 in the gaseous product; normal sulphates which form basic salts upon dissociation evolve only SO_2 when they pass from the normal to the basic state basic sulphates upon dissociation into MO set free only SO_3; metallic sulphates with bases M_2O_3 and M_2O do not form basic sulphates upon dissociation; the former set free SO_3, the latter SO_2.

Mendelejeff Table.

Group.	I.	II.	III.	IV.	V.	VI.	VII.	VIII.			
Series. 1.	. H 1			RH^4	RH^3	RH^2	RH	Composition of Hydrogen Compounds.			
2.	Li . 7	Be . 9.1	B . 11	C . 12	N . 14	O . 16	F . 19				
3.	. Na 23	. Mg 24.4	. Al 27	. Si 28	. P 31	. S 32	. Cl 35.5				
4.	K . 39.1	Ca . 40	Sc . 44	Ti . 48.1	V . 51.2	Cr . 52.3	Mn . 55	Fe 56	Co 59.1	Ni 58.5	Cu 63.3
5.	. (Cu) 63.3	. Zn 65.4	. Ga 69.9	. Ge 72.5	. As 75	. Se 79	. Br 80				
6.	Rb . 85.4	Sr . 87.5	Y . 89	Zr . 90.7	Nb . 93.5	Mo . 95.9	.	Ru 101.7	Rh 102.9	Pd 106.7	Ag 107.9
7.	. (Ag) 107.9	. Cd 112.4	. In 114.8	. Sn 119	. Sb 120.2	. Te 125.2	. I 126.9				
8.	Cs . 132.8	Ba . 137	La . 138.5	Ce . 141.5	Di ? 145	.	.				
9.	.	.	.	.	.	.	.				
10.	.	.	Yb . 173.2	.	Ta 182.8	W . 184	.	Os 190.9	Ir 193.1	Pt 195.2	Au 197.2
11.	. (Au) 197.2	. Hg 200	. Tl 204	. Pb 207.1	. Bi 208	.	.				
12.	.	.	.	Th . 232.4	.	U . 238.5	.				
	R^2O	R^2O^2	R^2O^3	R^2O^4 RO^2	R^2O^5	R^2O^6 RO^3	R^2O^7	Composition of Highest Saline Oxides. RO^4			

SUBJECT TO REVISION.

[TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS.]

Flameless Combustion.

BY CARLETON ELLIS, MONTCLAIR, N. J.

(Presented at a meeting of the New York Local Section of the Institute, Apr. 12, 1912.)

I. INTRODUCTION.

THE problem of the influence of hot surfaces upon gaseous combustion is one which, from a purely scientific stand-point, has engaged, for many years past, the attention of Prof. William A. Bone, of Leeds University and the Imperial College, London; and as his recent work has been the direct outcome of earlier scientific investigation, it will be appropriate, by way of introduction, to review briefly the present position of science with respect to this important subject, as stated by Professor Bone.

One may perhaps best arrive at an understanding of the term "flameless" or "surface" combustion by considering certain facts which differentiate it from the more familiar processes of combustion as they occur in ordinary flames. All hot surfaces have an accelerating influence upon chemical changes in gaseous systems. If, at any temperature, a gaseous system, *A*, tends to pass over into another system, *B*, contact with a solid at the same temperature will accelerate the process.

To take a very simple example, if a mixture of hydrogen and oxygen in their combining proportions (electrolytic gas) were maintained in an inclosure with smooth glass walls at a temperature of, say, 450° C., there would certainly be a tendency to form steam, but the rate of change would be negligibly small. If, however, there were brought into the system some porous solid material at the same temperature, so that a large surface was exposed to the gases, the rate of change would at once be rapidly accelerated in the layer of gas immediately in contact with the hot surface. Steam, the product, would diffuse outward from the surface, and the supplies of hydrogen and oxygen at the surface would be renewed by diffusion inward. Thus combustion would proceed heterogeneously at the surface until the transformation of the original electrolytic gas into steam was complete. In the circumstances just cited, the rate of combustion, although now quite measurable, would probably be insufficient to cause any self-heating of the inclosure. The temperature would remain at 450° C., which is well below the ignition-temperature of the combustible mixture, or the point at which a solid would attain even incipient incandescence.

It is therefore necessary to distinguish between two possible conditions under which gaseous combustion may occur, namely: (1) homogeneously, that is to say, equally throughout the system as a whole, at temperatures below the ignition-point, slowly and without flame, and at temperatures above the ignition-point, rapidly and with flame; and (2) heterogeneously, or only in layers immediately in contact with an incandescent surface ("surface" or "flameless" combustion). Other things being equal, the heterogeneous surface-combustion is a faster process than the normal homogeneous combustion of ordinary flames.

The influence of hot surfaces upon combustion at low temperatures seems to have occupied the attention of several chemists (Dulong and Thenard and, independently, Dobereiner, in France, Sir Humphry Davy, William Henry, Thomas Graham, Faraday, and de la Rive, in England) during the first third of the last century; but no one of these distinguished men succeeded in evolving a satisfactory theory of the phenomenon, nor, with the exception of the famous "Dobereiner lamp," was there any practical outcome of their efforts. In 1836, after a long but abortive controversy between Faraday and de la Rive, interest in the subject was dropped, not to be revived until recent years. Professor Bone's attention was first drawn to the subject during the course of an investigation on the combustion of hydrocarbons at low temperatures. The subject soon became so absorbingly attractive that he embarked upon what proved to be a long inquiry into the influence of a great variety of hot surfaces upon the combination of hydrogen and oxygen at temperatures below the ignition-point. The inquiry has also included other cases of slow combustion; and experiments now in progress in his laboratory will materially advance the science of the subject.

Professor Bone's experimental results justify the conclusion that the power of accelerating gaseous combustion at temperatures below the ignition-point is possessed by all surfaces in varying degrees, dependent upon their chemical characters and physical texture. Moreover, the "activity" of a given surface can be enhanced or diminished at will in a truly marvelous manner by previous special treatment. Thus, for example, in the case of the combination of either hydrogen or carbon monoxide with oxygen, in contact with a non-oxidizable metal or non-reducible oxide, the "activity" of the surface may be greatly stimulated by previous contact with the combustible gas, and, conversely, may be diminished by previous contact with oxygen. Again, there is abundant evidence that the actual surface-combustion is dependent upon a prior "absorption" (or condensation) of the combustible gas, and possibly also of the oxygen, by the surface. To what extent oxygen is involved, is not as yet perfectly clear. The "absorbed" (or condensed) gas becomes "activated" (probably "ionized," as the physicists would call it) by association with the surface. Finally, certain important differences have been established between ordinary homogeneous combustion and heterogeneous surface- or flameless combustion. Thus, for example, whereas the presence of water-vapor certainly accelerates, if it is not essential to, the homogeneous combustion of carbon monoxide, it greatly retards the heterogeneous combustion of the same gas in contact with a surface such as fire-clay. Again, whereas methane has, in ordinary flames, a much greater affinity for oxygen than either hydrogen or carbon monoxide, a hot surface, by virtue of some "selective" action, will completely reverse this usual order of things—a remarkable circumstance, than which no better proof could be afforded of the reality of surface-combustion.

In a discussion before the British Association in 1910, Sir J. J. Thompson insisted that combustion is concerned not only with atoms and molecules, but also with electrons—*i. e.*, bodies of much smaller dimensions and moving with very high velocities—and suggested that "in reference to the influences of hot surfaces in promoting combustion, to which Professor Bone has drawn attention, it was not improbable that the emission of charged particles from the surface was a factor of primary importance." Those who have followed recent developments of the corpuscular theories of electrical action will recall the experimental proof that incandescent surfaces emit enormous streams of electrons, traveling with high velocities; and the action of such surfaces in promoting combustion may ultimately be found to depend on the fact that they bring about the formation of layers of electrified gas, in which chemical changes proceed with extraordinary rapidity.

A distinguishing feature of the new processes employing flameless combustion is, that a homogeneous explosive mixture of gas and air, in the proper proportions for complete combustion (or with air in slight excess), is caused to burn without flame in contact with a granular incandescent solid, whereby a large proportion of the potential energy of the gas is immediately converted into radiant form. The advantages claimed for the new system are: (1) the combustion is greatly accelerated by the incandescent surface, and may be concentrated just where the heat is required; (2) the combustion is perfect with a minimum excess of air; (3) the attainment of very high temperatures is possible without the aid of elaborate "regenerative" devices; and (4), by reason of the large amount of radiant energy developed, transmission of heat from the seat of combustion to the object to be heated is very rapid. These advantages are so uniquely combined in the new system that the resultant heating-effect is, for many important purposes, not only pre-eminently economical, but also easy of control.

II. Diaphragm-Heating and Its Applications.

In this process the homogeneous mixture of gas and air is allowed to flow under slight pressure from a suitable feeding-chamber, through a porous diaphragm of refractory material, and to burn without flame at the surface of exit, which is thereby maintained in a state of red-hot incandescence. The diaphragm is composed of granules of fire-brick bound together into a coherent block by suitable means. Its porosity is graded to suit the particular kind of gas for which it is to be used; but for undiluted coal-gas, or coal-gas containing only a small proportion of water-gas, a diaphragm so porous that the gaseous mixture will readily flow through it at a pressure of $\frac{1}{8}$ in. water-gauge is employed. It is mounted in a suitable casing; the space inclosed between the back of the casing and the diaphragm constituting a convenient feeding-chamber for the gaseous mixture, which is introduced at the back. Such a mixture may be obtained in either of two ways: namely, (1) by means of suitable connections through a Y-piece with separate supplies of low-pressure gas and air (2 or 3 in. water-gauge only is sufficient); or (2) by means of an injector-arrangement connected with a supply of gas at 2 lb. per sq. in. pressure. In this case the gas draws its own air from the atmosphere in sufficient quantity for complete combustion; the proportions of gas and air being easily regulated by a simple device.

To start up a diaphragm, gas is first of all turned on and ignited as it issues at the surface; air is then added gradually until a fully aerated mixture is obtained. The flame soon becomes non-luminous and diminishes in size; a moment later, it retreats to the surface of the diaphragm, which at once assumes a bluish appearance; soon, however, the granules at the surface attain an incipient red heat, producing a curious mottled effect; and, finally, the whole of the surface-layer of granules becomes red-hot and an accelerated "flameless combustion" comes into play. All signs of flame disappear, and there remains an intensely glowing surface—a veritable wall of fire but without flame—throwing out a radiant heat which can steadily be maintained as long as required.

The actual combustion in the diaphragm is confined within a very thin layer ($\frac{1}{8}$ to $\frac{1}{4}$ in. only) immediately below the surface, and no heat is developed in any other part of the apparatus. While the front of the diaphragm is intensely hot, the back of the apparatus is so cold that one can lay the hand on it. The combustion of the gas, although confined within narrow limits, is perfect; for, when once the relative proportions of gas and air have been properly adjusted, no trace of unburnt gas escapes from the surface. Moreover, the temperature at the surface of the diaphragm can be instantly varied at will by altering the rate of feeding of the gaseous mixture; there is no lag in the temperature-response—a circumstance of great importance in operations where a fine regulation of heat is required. The temperature of a diaphragm working on a mixture of coal-gas and air, at a given rate of feeding, depends on whether or not the intense radiation from its surface is impeded; with a freely radiating surface, the temperature of a properly-made diaphragm may be maintained at any point up to about 850° C. (say 1,550° F.), according to the rate of supply of combustible mixture. A curious feature of the diaphragm is the freedom from back-firing at this or lower temperature. Even when an explosive gaseous mixture is passed through the porous wall at a velocity very much smaller than the normal speed of back-firing of the mixture, no explosion backward will occur. Such a plain diaphragm may be placed at any desired angle between the horizontal and vertical planes.

The diaphragm method is appliable to a variety of combustible gases. Coal- or coke-oven gas (either undiluted, or mixed with water-gas), natural gas, gasolene-air gas, carburetted water-gas, are all well suited in cases where unimpeded radiation is required. I have recently found compressed liquefied gas (Blau gas) to give satisfactory results. Also, Professor Bone has constructed and successfully operated plane diaphragms of all sizes up to 4 sq. ft. in area, and is able to vouch from experience that their durability and radiant power are unimpaired, even after long-continued use.

Incandescence Not Dependent on External Atmosphere.

A further important point with regard to diaphragm-heating is, that the incandescence of the surface in no way depends upon the external atmosphere. When once the diaphragm has become incandescent, and the proportions of air and gas supplied in the mixing-chamber at the back have been properly adjusted, the surface will maintain its incandescence unimpaired even in an atmosphere of carbon dioxide, nitrogen, or steam.

Applications of Diaphragm-Heating.

I need hardly point out the many obvious purposes to which "diaphragm-heating" may be applied. Broiling, roasting, toasting, are at once suggested; others will doubtless occur to you;—such efficient means of attaining radiant heat can hardly fail to find new industrial uses. The evaporation and concentration of liquids by means of radiant energy emitted from a diaphragm fixed in a horizontal plane above the surface of the liquid is readily carried out.

For example, the evaporation of a solution of sodium silicate (water-glass) is an operation which could not be satisfactorily performed by the ordinary means of heating the vessel by flame from below. By the new method, however, only the topmost layers of the liquid are heated; the radiant energy of the diaphragm is instantly transmitted to the surface of the liquid, where it is absorbed and utilized for the evaporation. The sodium silicate separates out as a skin on the surface of the liquid, it is then dried by the radiant heat, and at intervals the crust of dry sodium silicate may be skimmed off. In this way, we are not only able to evaporate the solution with a great economy of heat, but we are also able to complete the evaporation of highly concentrated solution much more easily than by means of heat applied from below.

III. INCANDESCENT SURFACE-COMBUSTION IN A BED OF REFRACTORY GRANULAR MATERIAL.

This process is applicable to all kinds of gaseous or vaporized fuels, and to a great variety of both small- and large-scale industrial heating purposes. It consists essentially in injecting through a suitable orifice, at a speed greater than the velocity of back-firing, an explosive mixture of gas (or vapor) and air, in their combining proportions, into a bed of incandescent granular refractory material, which is disposed around or in proximity to the body to be heated.

Fig. 1 shows the process as applied to the crucible-furnace. The crucible is surrounded by a bed of highly refractory granular material. The mixture of gas and air is injected at a high velocity through a narrow orifice in the base of the furnace, and as it impinges upon the incandescent bed, combustion is instantaneously completed without flame.

The seat of this flameless combustion is in the lowest part of the bed; and the burnt gases, rising through the upper layers, rapidly impart their heat to the bed, maintaining in it a high degree of incandescence. Fig. 2 shows a similar arrangement for the heating of a muffle-furnace, an arrangement which needs no further explanation.

It is obvious that this process is adaptable to many other furnace-operations, as, for example, the heating of retorts, annealing-furnaces, and the like. It is not essential that the bed of refractory material shall be disposed around the vessel or chamber to be heated; it may be equally well packed into tubes, or the like, traversing the substance to be heated. This latter modification is important in relation to the melting of metals or alloys which are fusible at temperatures below about 600° C., and also in relation to steam-raising in multitubular boilers. By this process, much higher temperatures are attainable with a given gas than by the ordinary methods of flame combustion without a regenerative system. In fact, we have found that with any gas of high calorific intensity (such as coal-gas, water-gas, or natural gas), the upper practicable temperature-limit is determined by the refractoriness of the material composing the chamber (*i. e.*, the muffle or crucible) to be heated, rather than by the possibilities of the combustion itself. In a crucible fired by coal-gas on this system we have readily melted Seger cone No. 39, which, according to the latest determination of the *Reichsanstalt* in Berlin, melts at 1,880° C. We can also easily melt platinum, showing the possibilities of the method in regard to high temperatures with gas-fired furnaces. Using air preheated to 500° C., with coal-gas, a temperature estimated at somewhat over 2,000° C. has been attained. A very resistant chromite, not melting at 1,880° C., was fused in this way. Crucibles of alundum are fused without preheating the air.

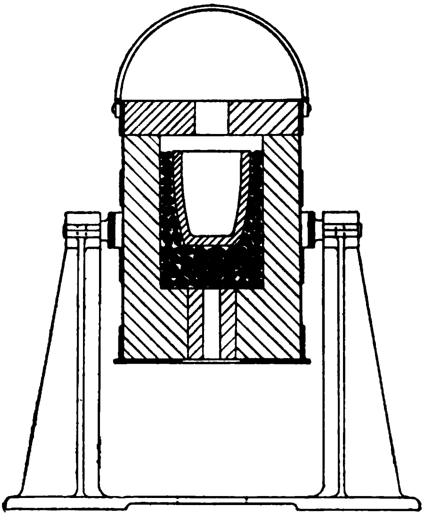


FIG. 1.—CRUCIBLE-FURNACE HEATED BY SURFACE-COMBUSTION.

For the very high temperatures obtained with coal-gas, water-gas, or natural gas, Professor Bone employs a bed composed either of fragments of magnesia, which has been burned at a high temperature, or of a neutral and highly refractory material specially prepared for this purpose. When the temperature required does not exceed 1,300° C., the bed of refractory material may be composed of a good quality of fire-brick, crushed and meshed to a suitable size. As already remarked, the method is applicable to all kinds of gaseous and vaporous fuels, but naturally the maximum temperature obtainable in any given case will depend upon the volume and heat-capacity of the products for a given heat-development in the bed. Thus, while with actual coal-gas, water-gas, or natural gas, it is possible to attain temperatures up to at least 2,000° C., about 1,500° C. would probably be the maximum temperature obtainable without regeneration with producer-gas of low calorific intensity, such as Mond gas. With some degree of heat-recuperation, which in such a case would be quite practicable, this limit could be in all probability considerably exceeded.

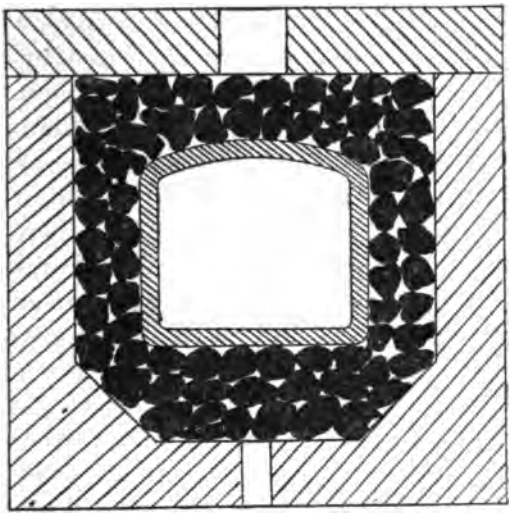


FIG. 2.—SURFACE-COMBUSTION APPLIED TO A MUFFLE-FURNACE.

The following are the results of a test on a muffle-furnace in which the muffle was heated between 815° and 1,425° C., with fully aerated coal-gas.

Results of Test on a Muffle-Furnace.

Dimensions of muffle, 9.5 by 5.25 by 3.25 in.

Temperature in Middle of Muffle.		Gas-Consumption to Maintain Temperature Constant.	Temperature of Product.	
Degrees C.	Degrees F.	Cubic Feet Per Hour, at 15°C.	Degrees C.	Degrees F.
815	1,499	21.0	540	1,004
1,004	1,840	35.3	645	1,195
1,205	2,201	58.0	870	1,598
1,424	2,596	79.0	1,085	1,985

Mean net calorific value of gas = 540 B.t.u. per cu. ft. at 15° C.

The conditions under which the tests were carried out made possible the accurate determination of the rate of gas-consumption requisite to maintain the muffle at any constant temperature between 815° and 1,425° C.

The temperatures given in the first two columns are those recorded by a standard thermo-junction placed in the middle of the muffle. The temperatures of the escaping products were also ascertained by means of a standard thermo-junction. It will be observed that the temperature of the products is in every case some 300° to 400° C. lower than that of the muffle. Even with a muffle-temperature of 1,424° C., there was no appearance of flame whatever at the top of the furnace. The gas-consumptions recorded in the middle column are extremely economical in comparison with ordinary heating by flame-contact. Thus, for example, in a similar test with a muffle of the same size, heated by flame-contact in a furnace of modern design, the gas-consumption to maintain the muffle at 1,055° C. (the maximum temperature obtainable) was 105 cu. ft. per hour; whereas by interpolation in the above table the consumption in the surface-combustion furnace at the same temperature would have been about 43 cu. ft. per hour only.

In a test which I witnessed, the consumption of gas by the Bone muffle-furnace was just one-half of that of a good type of the ordinary muffle-furnace of similar capacity, both being maintained for several hours at 1,500° C.

IV. SURFACE-COMBUSTION AS APPLIED TO STEAM-RAISING IN MULTITUBULAR BOILERS.

It is well known that hitherto the gas-firing of steam-boilers has not been very successful, either in thermal efficiency or in the rate of evaporation. All the gases used in this country for raising steam, such as blast-furnace gas, the surplus gas from by-product coke-ovens, natural gas, and producer-gas of various compositions, have been found amenable to the surface-combustion system. An eminent English blast-furnace engineer estimates that the efficiency of the best type of water-tube boiler, fired by blast-furnace gas, does not exceed about 55 per cent. Professor Bone asserts that careful observations, made on a battery of Lancashire boilers, fired by blast-furnace gas, evaporating water previously softened to within 4 degrees of hardness, with an attachment of the most approved type of economizers (so that the temperature of the burnt gases going to the chimney was reduced to the lowest possible point consistent with good draft), proved that the thermal efficiency did not, under the best of conditions, exceed 60 per cent. For boilers fired by coke-oven gas, one can safely say that the average thermal efficiency does not exceed 65 per cent., while in exceptional cases it may amount to perhaps 70 per cent.

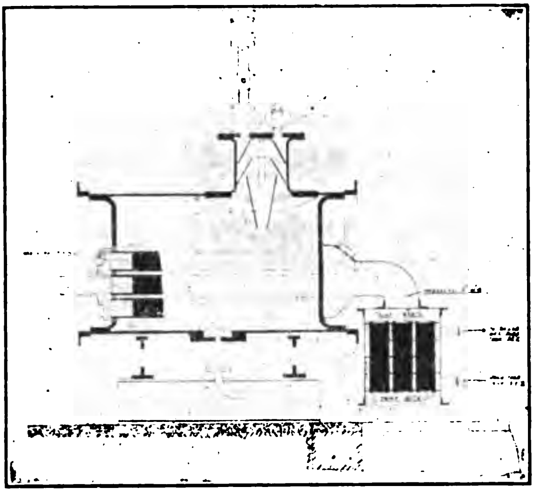


Fig. 3.—Experimental Boiler Operated by Flameless Combustion.

Fig. 3 represents a multitubular boiler of cylindrical section, operated by flameless combustion. It is traversed horizontally by a series of steel tubes, each 3 ft. only in length and 3 in. in internal diameter. These tubes are packed throughout with fragments of a suitable refractory material, meshed to the proper size. Into the front end of the tube, where the gaseous mixture is introduced, is fitted a fire-clay plug, through which is bored a circular hole about 0.75 in. in diameter. This plug serves the double purpose of keeping the front end of the boiler cool, and of providing a suitable aperture through which the gaseous mixture may be introduced at a speed much higher than the point of back-firing.

Attached to the front end of the boiler is a mixing-chamber of special design, not shown in detail in the figure. The mixture fed into the boiler-tubes from this chamber consists of the combustible gas, with a proportion of air slightly in excess of that required for complete combustion. The mixture is injected by pressure, or drawn by suction, through the orifice in this fire-clay plug, upon the incandescent material in the tubes. The combustion of the mixture in contact with the incandescent material is complete before it has traversed about 6 in. of the tube from the point of entry. The result is that the core of the material at this part of the tube is maintained at a high temperature, although the areas of actual contact between the hotter material and the walls of the tube are so rapidly cooled by the transmission of heat to the water in the boiler, that they never attain a temperature even approaching red-heat.

The combustion having been completed, the remainder of the material acts as a baffle towards the burnt gases as they traverse the tubes at a high velocity, causing them to impinge repeatedly on the walls of the tubes. The usual rate at which the gaseous mixture is fed into the boiler corresponds to the hourly consumption of about 100 cu. ft. of coal-gas plus six times its volume of air for every tube of the boiler, or an equivalent volume (*i. e.*, equivalent as regards heating-capacity) of any other gaseous mixture. Thus, for the ten-tube boiler on which the original experiments were made, the consumption of coal-gas was about 1,000 cu. ft. per hour, plus about 5,500 or 6,000 cu. ft. of air. These figures indicate the extremely rapid rate at which the mixture is caused to traverse the tubes.

Utilization of the Heat in the Exit Gases.

After the burnt products have traversed the boiler-tubes, their temperature is never more than about 70° C. above that of the water in the boiler (which, of course, depends upon the pressure at which the steam is being generated). This is a much lower temperature than that at which the products of combustion usually pass away from a multitubular boiler. But, in order to increase still further the output of steam, the products are passed through a short tubular feed-water heater, constructed on the same principle as the boiler. During a test carried out in Leeds, in which steam was generated at 100 lb. above atmospheric pressure, the temperature of the products leaving the boiler-tubes was 230° C.—the actual boiling-point of the water being 170° C. These products, still containing a certain amount of valuable heat, were passed through a feed-water heater, only 1 ft. long, containing nine tubes, of the same diameter as those in the boiler, packed with granular material. The hot products, continually baffled in their passage through the tubes, readily imparted their heat to the cold feed-water surrounding them; and their temperature was thereby reduced to somewhat less than 100° C.

The Ten-Tube Experimental Boiler.

The connections to the front of this boiler consisted essentially of a tube for the supply of gas, and another for the supply of air. The gas and the air were mixed before entering the feeding-chamber attached to the front plate of the boiler; the gaseous mixture was burned in the tubes of the boiler; and the products passed outward at the other end into a small chamber, and thence into the feed-water heater.

The mixture of gas and air was passed into the feed-chamber of this boiler at a pressure of 17.3 in. water-gauge. This pressure was necessary in order to overcome the resistance of the packing of the tubes. The pressure of the products entering the tubes of the feed-water heater was 2 in. water-gauge, so that the pressure necessary to force the gas through the zone of combustion, and thereafter through the remainder of the boiler-tubes, was about 15 in. water-gauge. In carrying out the test the water was evaporated at 100 lb. above atmospheric pressure; the temperature of the boiling water was therefore 168° C. (or 337° F.). The temperature of the combustion-products leaving the boiler-tubes was 230° C. The average temperature of the products leaving the feed-water heater was 95° C., or 203° F. The temperature of the water entering the feed-water heater was 5.5° C. (or 42° F.), and it was heated to

58° C. (or 136.4° F.) before entering the boiler, entirely at the expense of the burnt gases.

The ratio between the heat transmitted to the water and the net heat of combustion of the burnt gas in the boiler was 0.94; *i. e.*, over 90 per cent. of the heat generated was utilized.

It is one of the prominent merits of the new system that the gas is burned completely with a minimum excess of free oxygen. During the test in question, the average proportion of carbon dioxide in the combustion-products was as much as 10.6 per cent., while the oxygen was as low as 1.6 per cent. The most careful examination of the products failed to reveal the presence of the slightest trace of carbon monoxide, hydrogen, or methane. Therefore, the remainder of the gas was simply nitrogen. Even with as little as 0.5 per cent. of oxygen in the products, the combustion of the gas in the tubes is perfect, not a trace of combustible gas escaping.

The Skinningrove 110-*Tube Boiler for Coke-Oven Gas.*

That the gas-firing of boilers according to the new system has been advanced beyond the merely experimental stage is proved by the recent erection by the Skinningrove Iron Works Co., Ltd., Chuland, Yorkshire, of a 110-tube boiler capable of evaporating not less than 5,500 lb. of water per hour, fired by gas from a new installation of coke-ovens adjacent to the blast-furnace. As shown in Fig. 4, this boiler is a cylindrical drum 10 ft. in diameter, and only 4 ft. from front to back; it is traversed by 110 tubes of 3 in. internal diameter, packed with fragments of fire-brick. It is worked under the suction of a fan. To the front is attached a device whereby gas at 2 in. water-gauge pressure from a suitable feeding-chamber, together with a proper proportion of air from the outside atmosphere, is drawn (under the suction of the fan) through a short "mixing-tube" into each of the 110 tubes of the boiler, where it is burned without flame, in contact with the incandescent granular material. The products of combustion, having traversed the 4 ft. of packed tube, pass outward into a semicircular chamber at the back of the boiler, and thence through a duct to the tubular feed-water heater, represented in Fig. 5. A fan attached to the feed-water heater removes the cooled products and discharges them through a short duct into the atmosphere outside the boiler-house.

In construction, nothing could be simpler or more compact than a cylindrical shell only 4 ft. long by 10 ft. in diameter, supported on a casting and requiring neither elaborate brick-work setting nor chimney. The boiler has the further structural advantage over all other multitubular boilers, that the front plate can never be heated beyond the temperature of the water, however much the firing may be forced. This circumstance, coupled with the extremely short length of the tubes, implies an absence of strain and greatly reduces the risk of leaky joints. Another feature of the boiler which makes for efficiency is the steep "evaporation gradient" along the tubes. Under the normal working-conditions, the "mean evaporation" exceeds 20 lb. per sq. ft. of heating-surface, or about twice that of a locomotive boiler. Of the total evaporation, no less than 70 per cent. occurs over the first third of the tubes, 22 per cent. over the next third, and about 8 per cent. over the remainder. Such a steep gradient causes a considerable natural circulation of the water in the boiler, a factor of great importance in good working. As to thermal efficiency, it seems reasonable to expect that a boiler-unit which, while

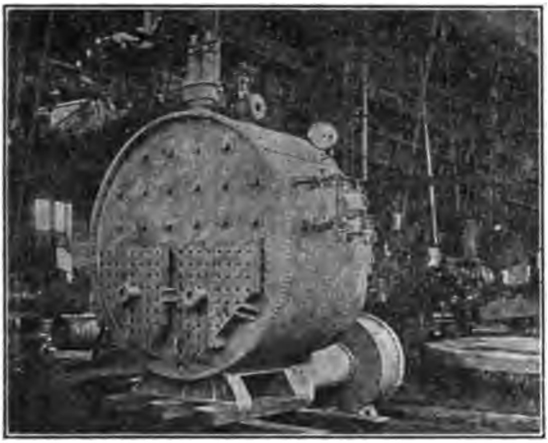


FIG. 4.—SKINNINGROVE GAS-FIRED BOILER OPERATED BY FLAMELESS COMBUSTION.

evaporating 20 lb. of water per sq. ft. of heating-surface, transmits upward of 90 per cent. of the net heat of combustion of the gas to the water, and which, if need be, can be forced to a 50 per cent. higher "duty," with only a slight drop in efficiency, will stand unrivaled as a steam-raiser. Moreover, in the case of a large boiler of (say) 100 tubes, "elasticity" maybe conferred by arranging the tubes in groups, so that they may be fired up or completely shut off, group by group, successively, in correspondence with variations in the load.

The Skinningrove boiler has proved almost completely automatic in its working, according to the statement of the manager of this plant, who says also: "The boiler has been off for the inspection of the tubes, which prove to be clean and free from scale, a fact which I attribute to highly rapid ebullition. During the length of time the boiler has been at work, we have had no trouble with priming; the steam having been at all times perfectly dry. The average temperature of the waste gases leaving the plant has been from 78 to 80° C., which is ample proof of the boiler's efficiency."

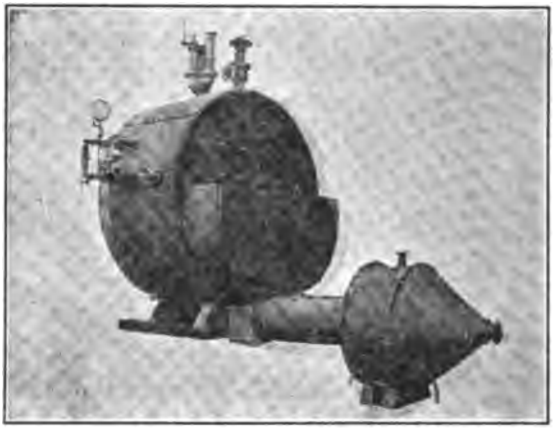


FIG. 5.—FEED-WATER HEATER OF THE SKINNINGROVE BOILER.

Experiments to determine the value of this type of steam-generator as a waste-heat boiler show important economies. Work on oil-fired boilers has also been carried out with satisfactory results.

V. THE MELTING OF EASILY FUSIBLE METALS AND ALLOYS.

It will be readily understood that the principle embodied in the boiler is capable of great extension. Thus, for example, it can be applied to (1) the preliminary concentration of dilute solutions and the heating of liquids generally; (2) the heating of large volumes of air; and (3) the melting of easily fusible metals and alloys.

I will here refer briefly to some experiments on the fusion of metals. Professor Bone's attention was first drawn to this subject by experts of one of the London gas companies, who represented that there would be a large field of usefulness for the process in melting type-metal for large newspapers, which require for their machines a continuous supply of molten type-metal. Fig. 6 represents an iron tank, efficiently lagged and filled to the top with molten lead at a temperature of, say, 50° above its melting-point. In the molten bath is fixed an iron tube, 2 or 3 ft. long and 3 in. in internal diameter. The tube is packed (like one of the boiler-tubes) with a suitable granular refractory material, and there are suitable arrangements for the introduction of the explosive mixture of gas and air which is to be burned in the tube. When once the device is started up, it can be worked continuously for days together. Solid lead is

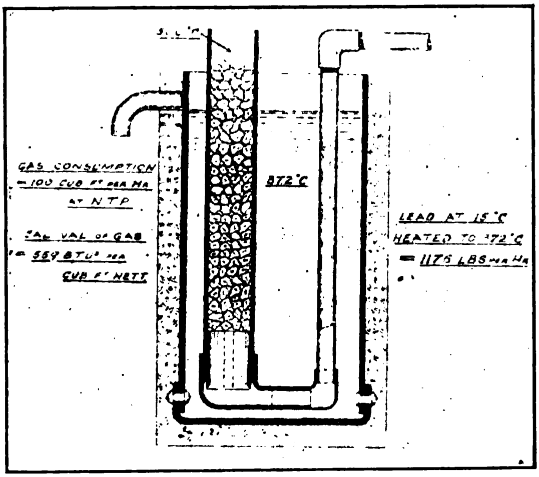


FIG. 6.—TANK FOR MELTING METAL, HEATED BY FLAMELESS COMBUSTION.

continuously fed into the apparatus, and the molten metal is allowed to run over through the spout indicated in the diagram. Experiments have been carried out with tanks holding up to 8 tons or more of molten metal, in which a series of combustion-tubes are fixed. By means of such an apparatus, lead (or other fusible metals or alloys) may be melted not only very rapidly, but with extraordinary efficiency. The following is the result of a test carried out at the experimental station, with a single-tube apparatus:

Lead-Melting Test.

	Degrees C.	Degrees F.
Temperature of metal charged, ...	15	60
Temperature of metal tapped, ...	372	682
Temperature of gases leaving apparatus, . . .	500	932

Lead melted per hour = 1,176 lb.

Heat required per hour to raise metal from 15° C. to 372° C. = 1,176 × 32.67 = 38,420 B.t.u.

Gas burnt per hour = 100 cu. ft. at N.T.P.

Net calorific value of gas = 559 B.t.u. per cu. ft. at normal temperature and pressure.

$$\text{Ratio} = \frac{38.420}{55.900} = 0.686.$$

The conditions were so arranged that the mean temperature of the molten metal in the apparatus was 372° C. throughout the test. Lead ingots, each weighing about 30 lb., were added at intervals of 1.5 min., and the molten metal displaced was simultaneously run off into molds. Great care was taken to keep the bath thoroughly molten, and at a temperature within a few degrees of the mean value. Burning gas of net calorific value of 559 B.t.u. per cu. ft., at the rate of 100 cu. ft. per hour, it was found possible to raise the temperature of 1,176 lb. of lead per hour from 15° to 372° C.; the temperature of the products of combustion leaving the tube being constant at 500° C., or only 128° above the temperature of the molten metal. Using the latest determination by Spring of the specific heat of lead at temperatures up to and above its melting-point, and adopting the usually accepted value for the latent heat of fusion of lead, Professor Bone estimates that at least 70 per cent. of the heat developed was utilized. My observations of this type of heating-apparatus, in comparison with externally heated melting-pots, show a great difference in fuel-consumption in favor of internal heating.

Many other applications of flameless combustion are undergoing exhaustive investigation by Professor Bone and his able colleague, C. D. McCourt; and in the near future we may look for further interesting developments within the many departments of the field of combustion without flame.

DISCUSSION.

ARTHUR H. ELLIOTT, New York, N. Y.: We have all been very much interested in the results of Dr. Bone's work, which Mr. Ellis has presented to us this evening. The high temperatures attained by this method perhaps afford us an explanation of the temperatures reached in the puddling-furnace with its crude and almost primitive methods of burning fuel, by means of which we have, nevertheless, been able to melt iron in a comparatively short time, and keep it in a molten condition down to a high degree of purity. No doubt many of us have wondered, as I have done, how such a result could be obtained in such a way; and it occurs to me that, perhaps, Dr. Bone has supplied the explanation in disclosing the effect on combustion of burning gas and air after passing through, and in contact with, a porous refractory body. May not the fettling of the puddling-furnace have, in some degree, an effect upon the combustion like the influence of the porous diaphragm employed by Dr. Bone?

C. A. MEISSNER, New York, N. Y.: I have been much interested to observe the shortness of the tubes of the boiler and of the hot-water heater shown by Mr. Ellis, especially in connection with his statement that these tubes showed no deposit of scale around the point of combustion, and no deterioration. In view of the quick and complete combustion of fuel under the Bone system, and therefore the strong concentration of heat at one point, these facts, if borne out, present some very interesting features.

ANTON EILERS, New York, N. Y.: I should like to ask Mr. Ellis if any experiments have been made in the direction of applying flameless combustion in large metallurgical operations, especially as to its possible use in the very large reverberatories used in copper-smelting. If not, I would suggest that experiments in this direction might be expected to be very important.

WILLIAM KENT, Montclair, N. J.: Mr. Ellis's paper contains many interesting facts; but I do not think it is necessary to have a new theory of "surface combustion," increased chemical capacity, or catalytic action of the surfaces to explain the phenomena that he describes. Whenever a combustible gas, such as methane, is intimately mixed with air in the right proportions, so that each particle of gas is in intimate contact with enough oxygen for its complete combustion, and the mixture is heated at any point to the igniting-temperature, it will burn with the rapidity and completeness of an explosion. No kind of combustion can be more perfect, and it cannot be improved by contact with the surface of any solid material. The porous diaphragm shown by Mr. Ellis merely acts like the fine wire mesh of the Davy safety-lamps; that is, it prevents the retreating of the flame back into or behind the diaphragm so long as the latter is below the igniting-temperature, but as soon as the diaphragm gets heated on the flame side to the igniting-temperature, the flame will retreat and burn in the rear of the front surface, and among the particles of the diaphragm itself.

In regard to the rapid rate of transmission of heat through boiler-tubes when they are filled with sand, among the particles of which the gas is burning, the rapidity of transmission is not due to any new kind of combustion, but merely to the heat-conducting power of the sand in direct contact with the metal of the tube. The sand constitutes an "extended surface," and it acts precisely like the projections of the inner surface of the Serve tube which was used to some extent in English boilers several years ago, or like the spiral metal "retarders" in contact with the inner surface of the boiler-tube, which are occasionally used in fire-tube boilers.

It is not conceivable by any one familiar with steam-boiler practice that the Skinningrove form of boiler can ever come into common use. While the rate of transmission of heat through the tube-surface may be from two to four times that in ordinary boilers, the cost of the boiler will be more than four times as great as that of a water-tube boiler of the same extent of heating-surface, or a water-tube boiler of four times the heating-surface can be built for less money. Besides this, the cost of power to push the gases of combustion through the sand-filled tubes would be an insuperable objection to the commercial use of such a boiler.

Charles Edward Lucke, New York, N. Y.: It was not my intention in coming here to speak of my own work in this field, but rather to learn of any new developments that might be reported by the lecturer of the evening. However, it might be well to review some of my old experiments and apparatus in the burning of explosive gaseous mixtures that seem to disclose all that has been described as the work of Professor Bone, and to offer a more simple and adequate explanation of the phenomena. Some years ago (about 1900), the possibility of operating internal-combustion engines by a different plan from that used by Otto and Diesel was subjected by me to a somewhat lengthy study. The scheme that seemed to offer many advantages consisted in compressing air, burning fuel in the compressed air at constant pressure, and subsequently expanding the heated products of combustion, a plan which, of course, would include the gas-turbine as one form. It became clear very early that, to get the maximum work from the volume of air compressed, this air should be made to combine with the maximum amount of fuel, or that the combustion should take place with air and fuel in combining proportions. With fuels in the gaseous or vapor form, such combustion would be explosive; and so it seemed necessary to investigate the problem of burning explosive gaseous mixtures continuously localized in a so-called combustion-chamber or furnace. Review of U. S. patents revealed many burner- or furnace-constructions intended for the purpose, and practically all of them were tried, as well as a good many others designed by me; but all failed to work properly. These failures led to a study of the literature of explosive gaseous combustion, and practically everything in print was examined, with a view to discovering some basic principles for the operation of the burner- or furnace-apparatus. With the help of this literature research, combined with the experimental experience so far gained, I developed a method of burning explosive mixtures, derived from both gas- and oil-fuel, in localized zones under perfect control, and the means I employed were similar to those described by Professor Bone and repeated by Mr. Ellis. It is most interesting, however, to note that the reasoning that led to the result, and the explanation of the phenomena, are different from what they offer; and, of course, it is natural that I should think my own the more satisfactory.

Great stress has been laid by Professor Bone on the catalytic agency of a porous incandescent surface itself, as a means of promoting combustion, and as an explanatory theory of the action taking place in the apparatus used by us. There can be no doubt of the accelerating tendency of hot bodies on the rate of combustion-reactions. Every one who has worked with fires, especially gas- and oil-fires, knows that more fuel can be burned in a furnace after it is heated up than when cold. But, as an explanation of the peculiarities of the explosive combustion processes, this is entirely inadequate. The true explanation, as I have previously pointed out, must be based on adjustment of velocities, velocity of flow to velocity of propagation of inflammation. When porous beds or plates of solid matter are used to affect or control this velocity adjustment, and become hot in so doing, of course, they then bring into play any catalytic or otherwise accelerating tendencies that belong to them; but the prime action still remains one of adjustment of flow-velocity to rate of propagation. In free flow-paths it is always true that the flame-cap or surface of combustion will localize or remain fixed only where flow-velocity is equal to rate of propagation, or at the place where the mixture approaches as fast as it can burn back towards the source of supply.

Let us consider the line of reasoning that leads up to this conclusion and furnishes the explanation of all these examples of explosive combustion, starting, first, with the burning of masses of explosive mixture at rest, and later with streams of mixture flowing through various forms of passages. Imagine Fig. 7 a mass of mixture at rest, ignited at the point *O*, and variously restricted by limiting walls; at *A*, no limit whatever; *B*, ignition at the apex of a solid corner; *C*, ignition on a flat plate; *D*, inside the apex of a cone; *E*, ignition at a point inside of a cone connected with a tube; and *F*, at the middle of a tube alone. In all cases the inflammation started at *O* will move at equal velocities in all directions and at a rate generally termed the rate of propagation and designated as *r* ft. per sec.

At the end of 1 sec., combustion will be taking place on the surface of a sphere whose radius is *r* ft.; and this spherical surface, whether complete or interrupted by any boundary-walls, will separate burnt from unburnt gases. Each successive second the spherical surface of combustion will increase in radius by *r* ft. until all the mixture has been consumed. These successive spherical surfaces will be equally distant when the rate of propagation (*r*) is constant; but should it increase, the

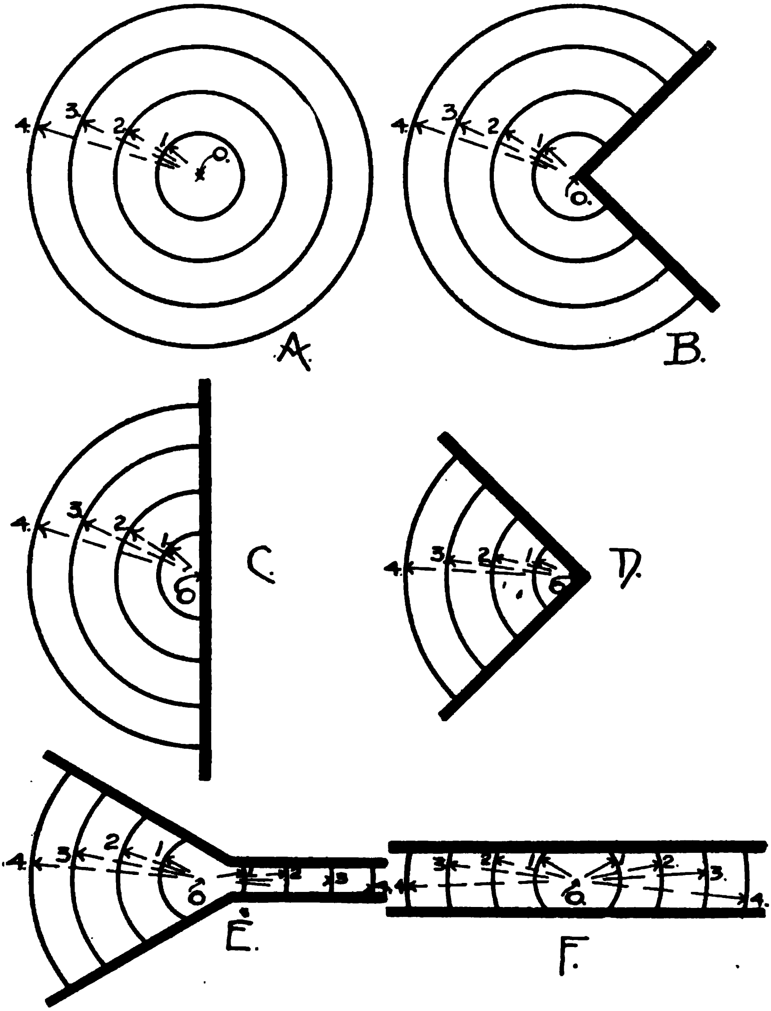


FIG. 7.—COMBUSTION-SURFACES WHEN EXPLOSIVE MIXTURES, VARIOUSLY CONFINED AND AT REST, ARE IGNITED.

distances between the spheres of each successive second will increase, and inversely for a decrease of *r*. There are many known causes of variation in the rate of propagation that need not be taken up here, except to note that the maximum rate is that of the detonating wave studied by Berthelot, Mallard, and Le Chatelier, and more recently by Dixon, while the minimum rate is always zero, and the normal rate for any mixture may be reduced to zero by sufficient cooling, or accelerated by heating.

However the rate of propagation may vary, the combustion will remain on a spherical surface if the mixture be homogeneous; but non-homogeneous mixtures, or mixtures in irregular chambers exerting variable influences on the rate of propagation, will burn on surfaces that differ from spheres only as these influences determine. The influences that control the velocity of propagation are, in the class of cases under discussion, due mainly to heating and cooling of the mixture; and these are illustrated in Fig. 8, showing the mixture at rest in tubes of various kinds. In the first tube, *A*, the walls are supposed to be absolutely non-conducting and of the same temperature as the mixture, so that the successive combustion-surfaces corresponding to each second of elapsed time after ignition at *O* are truly spherical and equidistant. Tube *B* is provided with a hot jacket on one end and a cold jacket on the other, affecting the temperature of the mixture next the walls. On the cold side, the edges of the spherical surface of combustion are bent back, due to cooling, while on the hot side they are bent forward, without affecting the distances between successive surfaces at the center. If the hot side be warmed to the temperature of ignition, of course the flame-travel will be wholly changed, as each point of the wall becomes then a new igniting-point, acting the same as *O*. Should the whole mass of mixture in the hot end be uniformly hot, and that at the cold end uniformly cold, then the successive surfaces would be unequally spaced, as indicated by the dotted lines, without bending at the angles. Tubes *C*, *D*, and *E* are non-homogeneous, having in the left-hand end packing of various kinds; *C*, a bundle of small tubes or rods; *D*, flat plates close together; and *E*, granular solid material. These all exert cooling influences on the flame passing between, and retard the whole advance, as is indicated by the lesser distances between the surfaces on the packed side, compared with the free side. In all these cases, if the gas-passages be small and the quantity of solid matter separating them be large, then the cooling influence may be so great as to prevent flame-passage at all, by reducing the rate of propagation to zero; and one common application of this principle is the wire gauze partition, illustrated in Tube *F*.

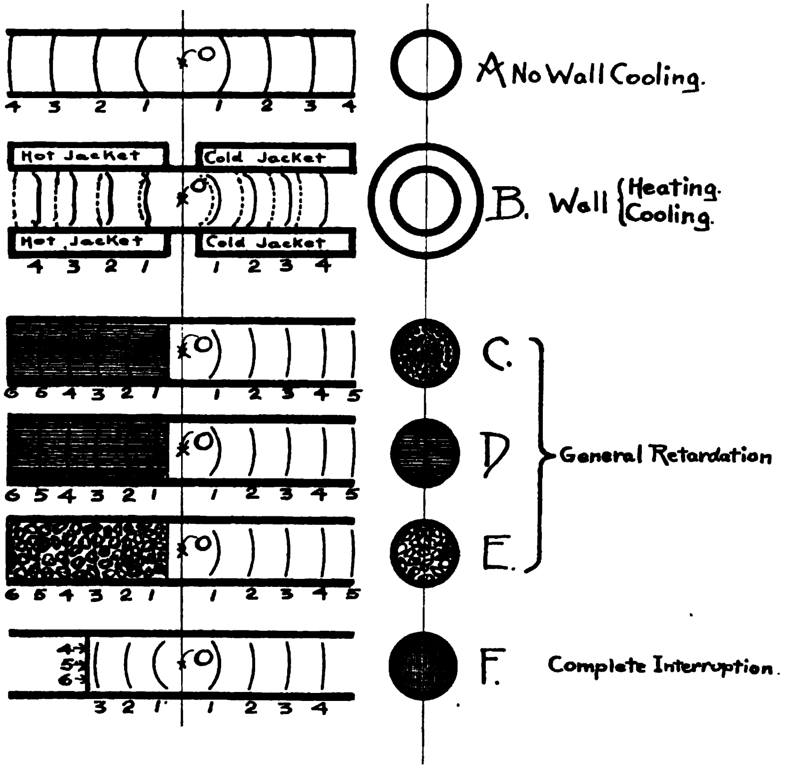


FIG. 8.—COMBUSTION-SURFACES WHEN EXPLOSIVE MIXTURE AT REST IS IGNITED.

Should the solid matter forming the multiple flow-paths for the passage of the flame be heated, it will exert an accelerating influence on the rate of propagation, and not only cease to be a flame-interrupting partition, but actually promote flame-passage.

With this preliminary discussion on the movement of the surfaces of combustion in quiet mixtures, at rates determined by heating and cooling influences in the flame-path, and the effect on the position and shape of the surfaces, let us consider an explosive mixture in motion, having a flow-velocity of v ft. per sec. as it issues from a single orifice, and subsequently

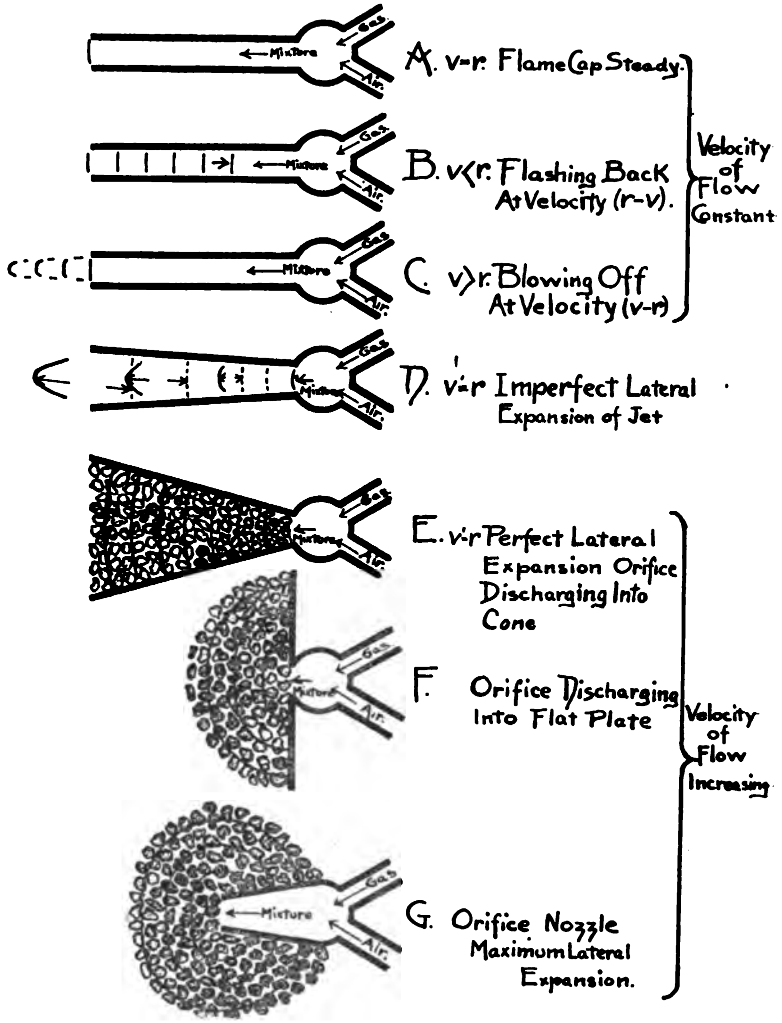

Mixture
Gas
Air
A. v=r. Flame Cap Steady.
B. v<r. Flashing Back At Velocity (r-v).
C. v>r. Blowing Off At Velocity (v-r)
Velocity of Flow Constant
D. v=r Imperfect Lateral Expansion of Jet
E. v=r Perfect Lateral Expansion Orifice Discharging Into Cone
F. Orifice Discharging Into Flat Plate
G. Orifice Nozzle Maximum Lateral Expansion.
Velocity of Flow Increasing

FIG. 9.—COMBUSTION-SURFACES WHEN EXPLOSIVE MIXTURE FLOWS THROUGH VARIOUSLY FORMED PASSAGEWAYS.

changing velocity, as illustrated in Fig. 9. In the first case, *A*, the mixture flows through a parallel-wall tube at a velocity, v, equal to the rate of propagation, r, so that, after ignition at the outlet, the flame-surface remains suspended in space, the mixture being supplied just as fast as it can burn. At *B*, the velocity of flow, v, is less than the rate of propagation, so that the combustion moves against the flow-current, or back-flashes, at a rate equal to the difference $r - v$, while at *C* the velocity of flow is greater than the rate of propagation and the surface of combustion moves with the current or blows off at a rate equal to the difference $v - r$. In blowing off, however, the jet of mixture, as it moves through the atmosphere, diffuses at the edges, so that the cross-section that remains explosive grows smaller, while all gas that is torn off by diffusion is lost for combustion entirely.

That the flame-cap or surface of combustion may be kept steady, or localized, requires very accurate adjustment of flow-velocity to rate of propagation, which, to be practical and useful, must be automatic. I, therefore, early adopted the broad principle of supplying mixture at a velocity greater than the rate of propagation, with subsequent reduction of flow-velocity by sidewise expansion. This was first tried as in *D*, where a conical nozzle was relied upon for exansion and reduction of velocity, but without complete success, because, except when the angle of the cone was very small, the jet expanded sidewise too slowly. It was expected, for example, that where the area of the cone was twice the area of the throat, the velocity of flow would be half that of the throat, so that if the throat-velocity were twice the rate of propagation equality would automatically occur at the point where the nozzle-area was twice that of the throat. Likewise, as the throat-velocity rose to three and four times the rate of propagation, the flame-cap would locate by automatic equalization between velocity and rate of propagation where the area of cone was three and four times the throat-area respectively. What actually happened is illustrated in *D*, somewhat exaggerated. The successive flame-caps due to increase of throat-velocity located beyond the expected position, and, becoming pointed, proved insufficient sidewise expansion. This was entirely cured, even for wide angles of cone, by filling the cone with loose refractory material, as in *E*. Of course, this cut down the area available for flow, leaving only so much as corresponded to the voids, which depend on the size of the grains, being less for small and more for large grains. This worked perfectly and called into play the secondary influence of acceleration of rate of combustion as soon as the material became hot, as it did almost immediately by the combustion proceeding between the particles.

By the automatic adjustment of flow-velocity and rate of propagation the combustion-surface instantly localizes between the granules and heats them rapidly to incandescence, and then, as the rate of combustion is increased by this high temperature, the equalization-zone moves back a little, just as it would with a flow-velocity reduction; but it soon reaches a steady state. Increase of flow-velocity will cause the combustion-zone to move away enough to conform to equalization, as before; and a great range of feed-rates is possible, if the angle of the wedge be large enough, without driving the combustion beyond imposed limits. An angle of 180° is illustrated in *F* and 360° in *G*. In all these cases, it matters not at all what refractory material is used to promote the lateral spreading of the mixture-stream, reducing flow-velocity and, by heating, accelerating the rate of propagation, to bring about equalization of velocities. Nor does it matter just how the material is formed, whether as oose or bonded granules, or holes in plates, or a series of baffles, or porous baked plastic matter. In my early work broken fire-brick was always used because of its availability; and, shortly after igniting, the granules always stuck together, forming a porous lump or plate or diaphragm, depending on the conditions ot forming; but the combustion of the mixture was exactly the same in this bonded mass as in the loose granules, except, of course, when, as occasionally happened, the whole mass really fused and flowed—an action which led to the subsequent use of more highly refractory material.

In the last form shown, Fig. 9, *G*, where the nozzle projects into the jet-spreader, the zone of combustion returns on the nozzle and will heat it above the temperature of ignition, causing the mixture to ignite inside, unless the walls be thick and good insulators. To prevent any such igniting of the mixture in feed-nozzles, it is generally best to use flows of the type *F*, which, of course, may be multiplied without end to enable any desired area of fire-bed to be developed. A variety of forms of such multiple feeds are shown in Fig. 10, all of which work precisely alike in principle. The first construction, *A*, has several feed-orifices, each supplied by a branch from a header-pipe, the

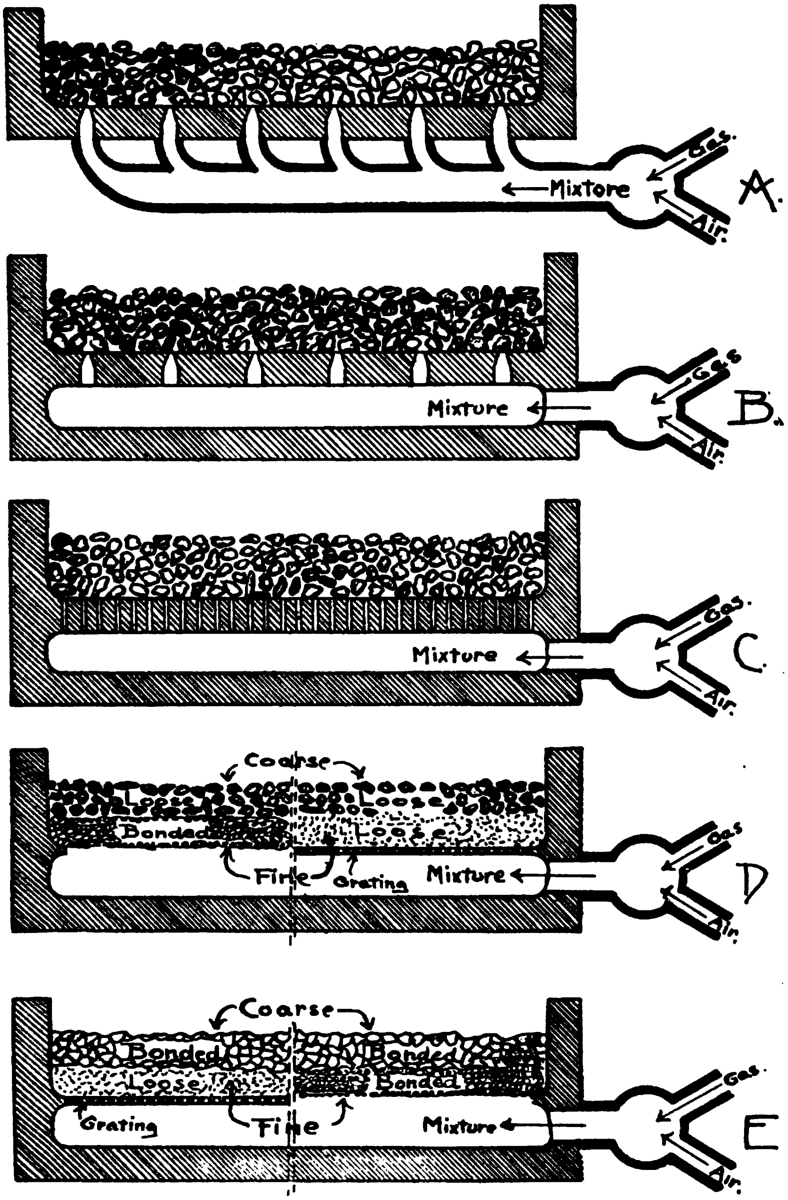


Fig. 10.—Various Forms of Porous Beds for the Combustion of Explosive Mixtures Flowing Through.

velocity at the exit of each into the bed being greater than the rate of propagation, and reducing, as it flows through the bed, by more or less spherical spreading. When the diameters of these combustion-spheres become greater than the distance between orifices, they join and produce a continuous wavy surface of combustion above, completely protecting the feed-orifices. Exactly the same result is attained by the holes in the partition shown in *B*, all fed from a common feed-chamber, and likewise in *C*, where the number of nozzles is multiplied considerably, reducing the size of the waves in the surface of combustion where the separate hemispheres join. In this latter case, *C*, the combustion-surface will locate closer to the orifice-plate and tend to heat it more than in case *B*—half the distance between centers of orifices approximately fixing the maximum height of combustion-surface. This makes it necessary, in using closely-spaced orifices, to form them in a refractory plate of sufficient thickness to prevent the supply side getting warmed to the ignition-temperature by conduction.

One convenient way of doing this is to form the feed-orifices by using a layer of fine granulated material under the coarse, as in *D* and *E*. This layer of fine material, if loose, is supported on a grating or perforated plate, through which the resistance is less than through the fine bed, so that the velocity will be high enough on exit from the fine bed; but if bonded or formed as a porous brick, the layer is self-supporting. The high exit-velocity from the fine or small-orifice bed prevents back-flash, and the reduction of velocity in passing into the upper area prevents blow-off. Four combinations of fine and coarse beds are shown: loose above bonded; loose above loose; bonded above loose; and bonded above bonded; it being understood, of course, that by bonded is meant any porous baked plastic matter, cemented granules, or just holes drilled or formed in plates, or a succession of baffles. When the lower or fine bed or its equivalent series of feed-holes is kept cool enough, and the holes small enough, it may act like an interrupting partition to resist back-flash, but it is impracticable to depend on this; the only really reliable dependence is sufficiently high flow-velocity through the orifices. It is interesting to note that the explosive combustion may be carried out for a small range by completely removing the top layer of *D* and *E*, which is exactly equivalent to removing the bed from the feed-orifices of *A*, *B*, and *C*, if the exit-velocity be just equal to rate of propagation or only slightly greater. In this case the action becomes the same as in Fig. 9, *A*.

It is believed that these principles, all of which were announced some years ago by me in papers before the American Society of Mechanical Engineers, are a clearer explanation of the working of Bone's apparatus than the speaker has offered, and considerably older.

SUBJECT TO REVISION.

[TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS.]

Electric Hoist at Hecla Mine, Burke, Idaho.

BY E. M. MURPHY, BURKE, IDAHO.

(Presented by invitation at a meeting of the Spokane Local Section of the Institute, Feb. 17, 1912.)

EIGHT years ago the Hecla mine, a lead-silver producer, situated at Burke, Idaho, was producing ore from but two levels, the 300- and the 600-ft. All hoisting was done with a 12- by 16-in. slide-valve, double-reel, hoist. The hoist was being worked to its capacity and was unsuitable for hoisting from greater depths. As the work of opening a new level was about to be started, it became necessary to remodel the then-existing equipment or install a new hoist. The recent advent of electricity from Spokane had made available a cheaper power than steam. By a series of tests conducted throughout the Cœur d'Alène district, it was found that it cost $109 per year to generate a horse-power with steam. Electric power was to be had for $50 a horse-power per year. It was decided to substitute for the engines a motor-drive, of sufficient capacity to operate from the 900-ft. level, and ultimately install an entirely new hoist. After disconnecting the engines, installing a new shaft, and making a few minor changes, a 300-h-p., 2,300-volt, 514-rev. per min., variable-speed motor of the induction type was geared to the reel-shaft. The reels were supplied with $\frac{3}{8}$- by 4-in. flat rope, and the duty of the hoist was about as follows:

Cage, 1,700; ore, 2,600; car, 1,400; rope, 2,250; total, 7,950 pounds.

The Bristol ammeter chart, Fig. 1, shows the combined current-consumption of the hoist- and mine-motors. The large intermittent load is clearly shown; this produced harmful fluctuations in the voltage of the long, lightly-loaded, 60,000-volt transmission-line from Spokane to the Cœur d'Alènes. The power company received no compensation for the increased cost of producing power for this intermittent or fluctuating load. In order to protect its system from similar excessive fluctuations of voltage in the future, the power company established a limit to the capacity of motors which it would permit to operate intermittently on its lines.

When planning for its new hoist, it was therefore necessary for the Hecla Co. to adopt an installation which would store energy during the periods when the demand for power is small, and return it during peak-loads. To meet this condition, it was decided to install an electric hoist of the Ilgner type, with the Ward Leonard control.

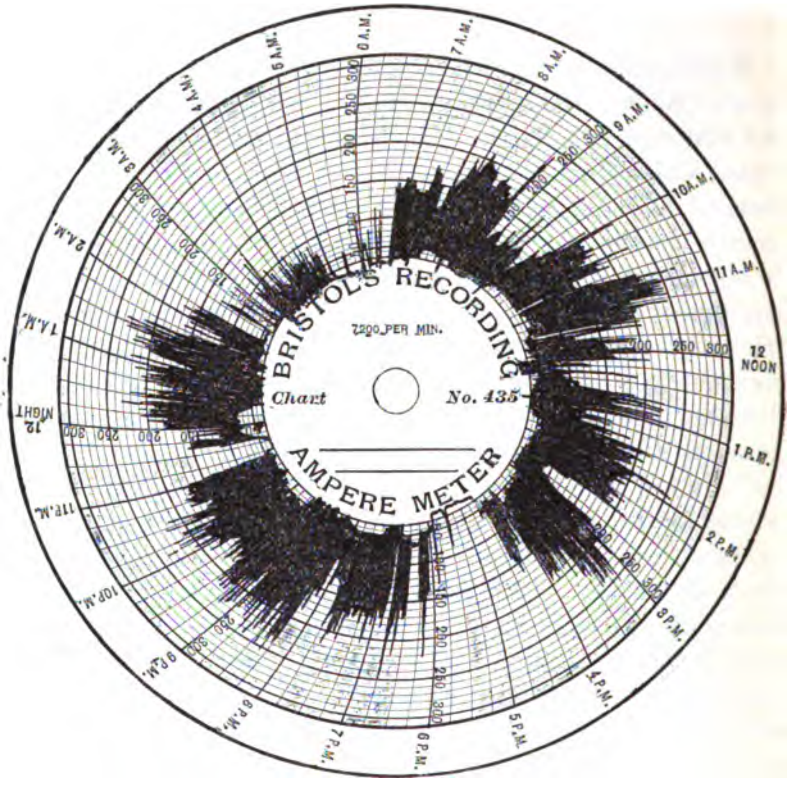


FIG. 1.—CURRENT-CURVE SHOWING LARGE PEAKS PRODUCED BY 300-H-P. INDUCTION-MOTOR OPERATING OLD GEARED HOIST.

The new hoist was installed during the spring of 1908, and began actual operations June 1 of that year. The equipment of the motor-generator set is as follows: The set is self-contained, having a cast-iron sub-base, four bearings and shaft; the driving-element consists of a 450-h-p., three-phase, 60-cycle, vound secondary motor, designed to operate between 2,000 ınd 2,300 volts.

The generator is a 450-kw., 525-volt machine, with commu- ation-poles to permit of its handling full-load current at any voltage between zero and maximum. Mounted on the shaft is fly-wheel, 7 ft. 9 in. in diameter, weighing 29,000 lb. This ly-wheel consists of a cast-steel spider, upon which is assem- oled a laminated steel rim. Upon the end of the shaft is nounted a 25-kw., 125-volt exciter, the armature of which is on the shaft of the set. The total fly-wheel effect of the set is 220,000 lb. at 1 ft. radius at 720 rev. per minute.

The hoist-motor is rated at 375 h-p., 500 volts, 60 rev. per nin., and weighs 51 tons. It is supplied with two bearings ınd is coupled to the reel-shaft through a forged flanged coup- ing. The reels are 5 ft., smallest, and 12 ft. 8 in., outside liameter. Each reel is fitted with a Lane clutch and a post orake. At present each reel carries 1,600 ft. of $\frac{5}{8}$- by 4.5-in. lat rope, this being sufficient to operate from the 1,200-ft. level. The center of the reels is placed 93 ft. 10 in. from the center of he shaft. Lack of available space, preventing the placing of he hoist further away, led to the adoption of flat rope. The heave-wheels are 8 ft. in diameter, and the centers are 102 ft. rom the collar of the shaft. The dump is 55 ft. 6 in. from the collar, and the skip-pockets are 25 ft. below the station at the 600- ınd 900-ft. levels, and 30 ft. below at the 1,200-ft. level. The kips are of 50 cu. ft. capacity, weigh 3,500 lb., and carry 2.6 tons of ore per trip. Double-deck cages hang beneath the kips at all times; each cage weighs 2,400 lb. The motor- generator set, the slip-regulator, and switch-board, are situated n a room adjacent to the hoist-room. On the switch-board are olaced the following instruments and equipment: one poly- ohase indicating wattmeter; one polyphase integrating watt- neter; one A. C. voltmeter; one A. C. ammeter; one double-read- ng D. C. voltmeter; one D. C. ammeter in the motor field; one D. C. voltmeter on the exciter circuit; one graphic recording D. C. wattmeter; one Tirril regulator for exciter voltage; one DPST switch in the armature circuit of the exciter; one auto- natic, hand-operated, three-phase, 2,300-volt, oil circuit-breaker; one generator-field switch; one exciter-field rheostat; one gen- erator-field rheostat; one motor-field rheostat; one switch in the magnet circuit of the slip-regulator switches; two line-relays for the slip-regulator; one compensator for line-relays; current and potential transformers for the various instruments.

Principle of Operation.

In order that the fly-wheel should give up part of its energy during the peaks, it is necessary to have some method of slowing down the set as the load comes on. This is accomplished by what is known as the slip-regulator. This device consists of a sectionalized resistance, comprising 13 sections, connected to the secondary of the induction-motor. Each section can be short-circuited by means of an oil-switch operated by compressed air. The air-valves are controlled by A. C. magnets wound for 50 volts, the magnets being supplied with power from a 2,300- to 50-volt transformer connected between the motor and its circuit-breaker. The magnet circuit comprises three circuits, viz., accelerating, drop-out, and hold-in; each of the first two has a switch in series, which is opened and closed by its respective relay. The relays receive power from a compensator which is excited from two current-transformers in the A. C. motor circuit. The relay switches can be set to operate at any desired value, within certain limits, by means of a dial-switch on the compensator, the one on the accelerating circuit being set to operate on a lower current than the one in the drop-out circuit.

With the A. C. circuit-breaker open, all the short-circuiting oil-switches are out, and the total resistance is in the secondary. To start the set, all that is necessary is to close the circuit-breaker; the starting current, being above that for which the relays are set, operates them, thus opening the accelerating and drop-out circuits. All the oil-switches, of course, remain open. As the set speeds up, the current diminishes and the relays close, causing an oil-switch to be pulled in. This short-circuits a section of resistance, the current increases, and the accelerating relay opens. As long as it remains open, the oil-switches remain out, with the exception, of course, of the one just closed, and this one remains closed, due to the action of the hold-in circuit. When the accelerating relay closes again, due to the fall in current, another oil-switch is closed. The opening and closing of the accelerating circuit, due to the increase and decrease of current, with its attendant closing of oil-switches, goes on until all the oil-switches are closed, and all of the resistance is cut out of the secondary. In hoisting, the reverse of the above goes on. As the load comes on, the relays open, and the last oil-switch to come in is the first to go out. This introduces resistance into the secondary, and the motor slows down, letting the fly-wheel take some of the load. Should the current rise again above the set value, another switch is dropped out; this operation can continue until nine switches drop out; four of the 13 are not connected to the drop-out circuit, and are used in starting only.

The hoist-motor is shunt-wound and is separately excited from the exciter. The current in the field can be adjusted by means of the rheostat in its circuit; when the proper adjustment is obtained, it is not necessary to change the setting. The motor is now operated with a field current of 90 amperes.

The Tirril regulator is seldom used to keep the exciter voltage constant, as the drop in voltage during balanced hoisting is found not to be objectionable. In unbalanced hoisting its use is essential, but the wear on the contacts is severe.

The control of the hoist-motor is effected through the field of the generator. The generator's armature is connected direct to that of the motor; any change or reversal in the generator voltage is accompanied by a similar change or reversal in that of the motor. The control resistance is situated under the hoistman's platform, and has a large number of points divided into four sections, connected up after the manner of the well-known Wheatstone bridge. In the off position of the controller a perfect balance exists, and no current flows in the field of the generator; the movement of the controller disturbs the balance, and this diverts voltage from the resistance circuit to the generator field. The controller is operated with a lever and quadrant similar to those on the brakes and clutches; moving the lever away from the operator starts the reels turning away from him, while a movement of the lever towards him starts the reels in that direction. In the generator-field circuit, and located back of the switchboard, is a rheostat for adjustment purposes; when once set, this needs no further attention. Mounted in front of the operator is a double-reading D. C. ammeter, which enables him to see the load on the hoist-motor at all times.

A diagram of connections as the equipment was originally installed is shown in Fig. 2, which does not, however, show the Tirril regulator and the D. C. graphic recording wattmeter. The exciter-field switch also is not shown, since it was installed after the hoist was in operation.

As originally equipped, the hoist had several devices, since discarded, which were intended to add to the safety of its operation and prevent abuse of the equipment by carelessness. A careless operator should not be tolerated on a hoist. The best safety-appliance on any machine is a careful manipulator. In connection with the controller was an automatic slow-down, designed to slow the hoist down as it approached the dump. Geared to the reels were two limit-switches, which controlled the opening and closing of a contactor. This contactor normally short-circuited a resistance in series with the generator field; within a certain predetermined distance of the dump, this contactor opened and the resistance was introduced into the generator field. The cutting in and out of this resistance was accomplished so suddenly that the commutation of the generator was affected. In addition, the failure of the apparatus to work at any time might have caused a serious accident, and it was deemed advisable to discard it.

A circuit-breaker was installed in the armature circuit of the exciter. Connected to this breaker, so as to open it, was an over-speed device on the motor-generator set, it being assumed that during unbalanced lowering excessive speed might be obtained, due to the pump-back action. In practice, this high speed is not developed. An over-load trip and an under-load trip, in the motor-field circuit, could also open the breaker. There are times when the failure of the D. C. power might prove disastrous and be attended with fatalities. It is essential to prevent any accidental opening of the circuit by using nothing but manually-operated switches. The breaker was removed and a switch installed. The over-speed device was connected so as to light a lamp, in front of the hoistman, in case the set speeded up. Should the occasion arise to cut the D. C. voltage off suddenly, it can be done by opening an exciter-field switch, installed in front and within easy reach of the hoistman. In Fig. 2, *X*, *Y*, and *Z* are the parts removed.

The hoist is also equipped with an air-operated device for

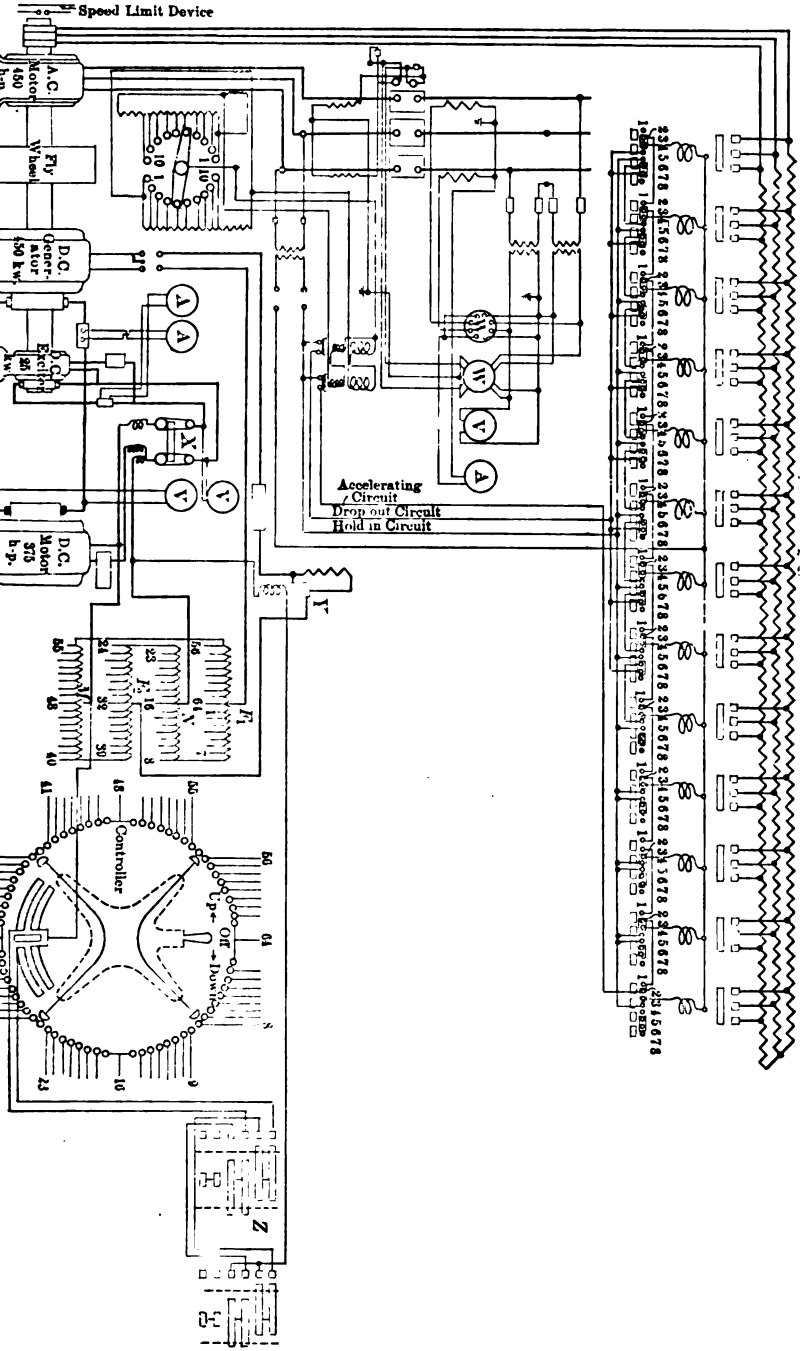


FIG. 2.—DIAGRAM OF CONNECTIONS.

throwing out the clutches and applying the brakes in case of an over-wind. Owing to the lack of sufficient head-room, this apparatus cannot be used. As a substitute, converging or "tight" guides are used, in addition to which are spring hooks, which engage the crosshead and prevent the falling of the skip, should the cable break after the skip reaches the sheaves.

Cost of Operation.

Before entering into the cost of operation of the hoist, an explanation of the present contract, signed in 1908, will serve to show on what basis a settlement is made for power consumed by it. The contract runs for a period of five years and is based on a maximum demand as well as a kw-hr. consumption. It will be noted that it penalizes a better combined power- and load-factor than 61 per cent. A power-factor of 100 per cent. and a constant voltage of 2,300 volts is assumed in all the calculations of maximum demand. In a contract calling for a maximum demand of 100 kw., a minimum sum of $335 ($3.35 per kw.) is paid each month; this sum entitles the consumer to 43,550 kw-hr. (130 for each dollar). At the same time, the maximum of 100 kw. must not be exceeded at any time during the month. In case more than 100 kw. is the maximum, the minimum bill is increased by $3.35 for each kw. in excess. For each dollar of this increase, the consumer is entitled to use 130 kw-hr. If the kw-hr. used exceed 43,550, the excess is paid for at the rate of $0.0112 per kw-hr. On the basis of 43,550 kw-hr. per month at $335 each kw-hr. costs $0.00769. This amounts to $50 per h-p. per year. The peak on the hoist never lasts 5 min., so the power never costs more than $50 per horse-power.

As the Hecla has but one hoist, the handling of all timbers, waste, etc., as well as the shifts, must be performed by it, in addition to the ore-hoisting. To give an idea of the work the hoist does, the following table was compiled for the period of time from Aug. 1, 1911, to Jan. 1, 1912:

Ore Hoisted.

	600-Ft. Level.	900-Ft. Level.	1,200-Ft. Level.
Skips,	2,241.0	9,968.0	9,961.0
Tons,	5,840.0	25,824.0	26,028.0
Monthly Average.			
Skips,	448.2	1,993.6	1,992.2
Tons,	1,168.0	5,164.8	5,202.6

Cars of Waste Handled.

	1,200 to 600.	Top to 600.	Top to 900.
Cars,	3,946.0	7,416.0	826.0
Average,	789.2	1,483.2	165.2

	Stulls.	Wedges.	Timbers Lagging and Chute.
	321,677 ft. B.M.	79 cars	475,140 ft. B.M.
Averages,	64,335.4	15.8	95,028.0

Power Consumed: 234,760 kw-hr., average equals 46,952 kw-hr., equals $361 per month. Total cost for power for each ton of output equals $0.0313.

In order to determine the cost per ton for power used during actual hoisting, a series of tests was taken, with the following results: From the 1,200-ft. level 32 skips (83 tons) were hoisted in 33 min., with a kw-hr. consumption of 142. On the basis of $0.00769 per kw-hr., the cost of hoisting the 83 tons was $1.092, or $0.0131 per ton. From the 900-ft. level 14 skips (36.4 tons) were hoisted in 11 min., with a kw-hr. consumption of 50, and a cost of $0.3854, or $0.0105 per ton. From the 600-ft. level 14 skips (36.4 tons) were hoisted in 10 min., with a kw-hr. consumption of 40, and a cost of $0.3076, or $0.00845 per ton.

To run the set light for 1 hr. requires 48 kw-hr., at a cost of $0.368. In service the set runs continuously during the 24 hr., with the exception of a period of about 4 hr. after midnight. After the power is cut off, the set will run for 1.25 hr., unless it is slowed down by hoisting, or the band-brake on the fly-wheel is applied. The hoist was guaranteed to maintain one-quarter output of the mine working unbalanced from the 2,400-ft. level (its ultimate depth). In order to test this feature, a load of 1,773 lb. was added to compensate for the extra weight of cable to 2,400 ft. This weight was obtained by placing a car with the required amount of ore in it on a cage-deck. The car was allowed to remain on the cage during the entire time of hoisting. Unbalanced hoisting was maintained at the rate of 11 trips an hr. from 900 ft. for 3 hr. All temperatures at the end of this time were well within the guarantees. Figs. 3 and 4 show 5-sec. readings for one complete trip during this test. In May, 1911, one of the clutch-arms broke, and the hoist operated unbalanced with entire satisfaction for a period of 20 hr., part of the hoisting being from the 1,200-ft. level.

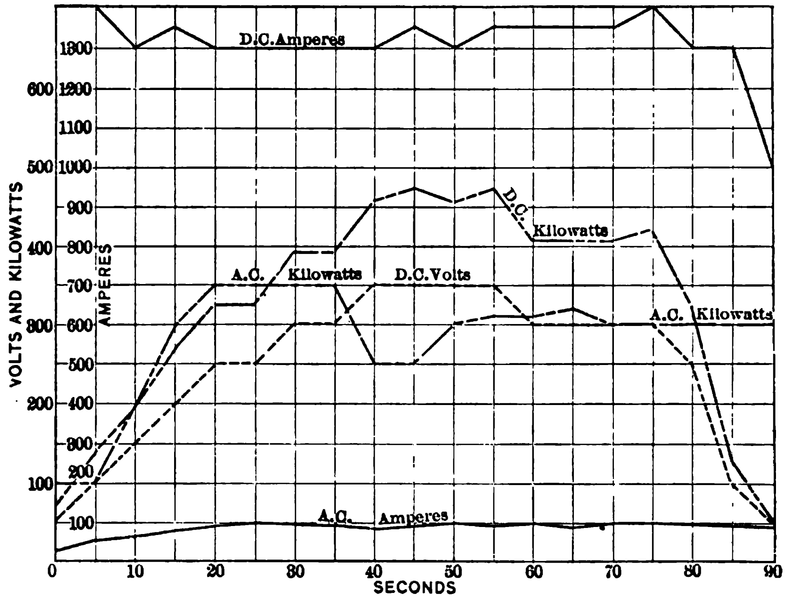


FIG. 3.—FIVE-SECOND READINGS TAKEN DURING UNBALANCED HOISTING TRIP FROM 900 LEVEL. SUFFICIENT EXTRA WEIGHT ADDED TO APPROXIMATE HOISTING-CONDITIONS FROM 2,400 LEVEL. RESISTANCE-SWITCHES OPERATE AT 310 KW. ALL RESISTANCE IN AT END OF TRIP.

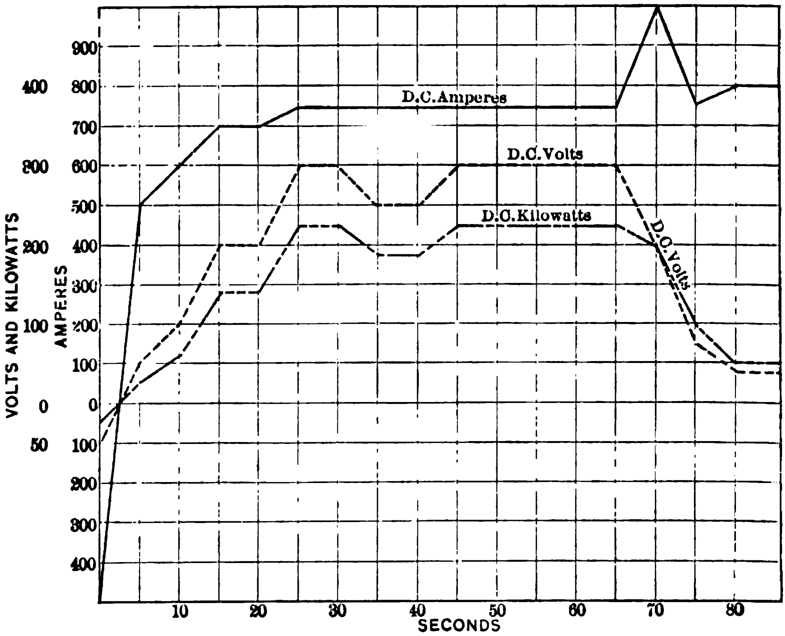
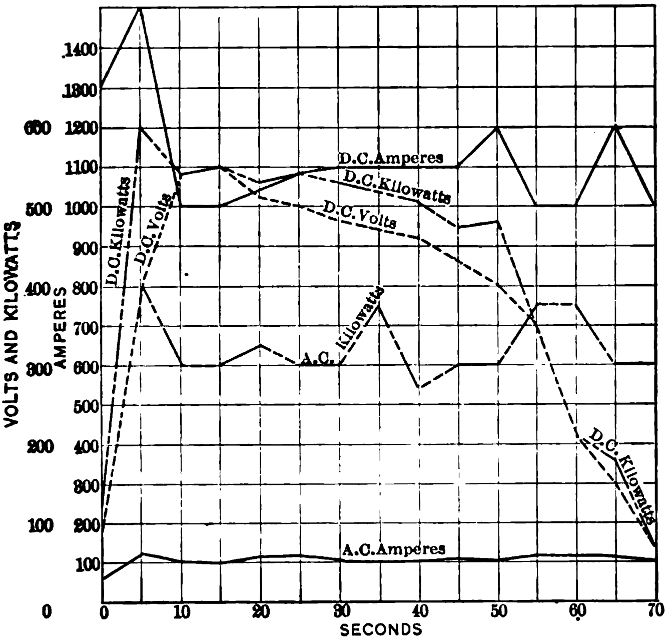


FIG. 4.—FIVE-SECOND READINGS TAKEN DURING UNBALANCED LOWERING, WITH EXTRA WEIGHT, TO 900 LEVEL.

Figs. 5, 6, and 7 show 5-sec. readings for unbalanced hoist-
ig from 1,200 and 900 ft. with ordinary load of 5,200 lb. of
re. Figs. 8, 9, 10, and 11 show 5-sec. readings of balanced
oisting from 1,200 ft. and 900 ft. with two different switch-
ettings. The effect of the different settings is clearly shown
ı the A. C. kw. input. It would seem from this that it would
e desirable to keep the setting that would give the minimum
put at all times. The objection to doing this is that the lower



G. 5.—Five-Second Readings for Unbalanced Hoisting from 1,200 Level Under Normal Conditions. Resistance-Switches Operate at 310 kw. All Resistance in at End of Trip.

ıe setting, the greater the number of switches that drop out,
ıd the slower the speed of the set becomes. When all nine
vitches drop out, it takes 55 sec. for them to come in again
ıd for the set to accelerate to full speed. It takes from 6 to
sec. to load and dump a skip, and if the speed in hoisting is
ot to be sacrificed, the switch-setting should be such that the
umber that drop out can come in again in the time it takes to
ad and dump. In practice this is found to be about three.

The setting, of course, has to be made for the lowest level hoisted from, or else the setting be changed for different levels.

The upkeep of the equipment for the three years and eight months it has been in service has been extremely low. The hoist-motor has needed no repairs, while the exciter has had but one new set of brushes. The generator requires about one new set of brushes a year. The A. C. motor has been the only source of expense, and like trouble could occur to any motor. Three times it has suffered a grounded coil during a lightning storm. The winding is a three-bank concentric winding, and replacement of coils is a tedious affair. A new set of collector-rings was also put on this machine last December.

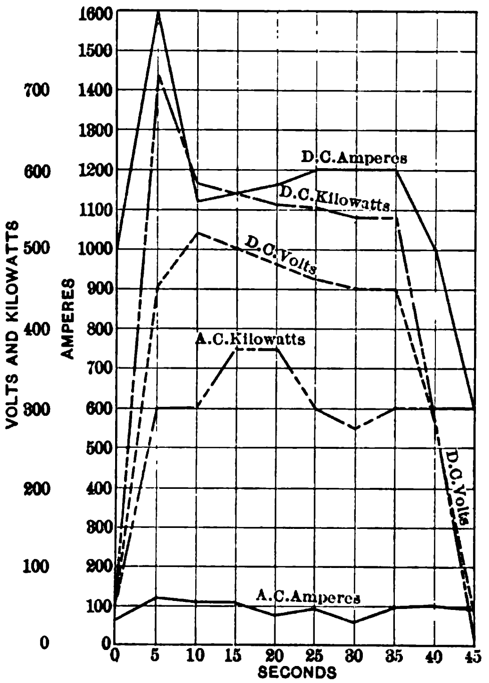


FIG. 6.—FIVE-SECOND READINGS FOR UNBALANCED HOISTING FROM 900 LEVEL UNDER NORMAL CONDITIONS. RESISTANCE-SWITCHES OPERATE AT 310 KW. ALL RESISTANCE IN AT END OF TRIP.

One noticeable feature of this installation is its noiseless operation and the facility of its control. The motor can be reversed in an incredibly short space of time, and during load-ng and dumping the brakes are seldom applied. The wear of he brake-blocks is inappreciable, and the original set will no loubt last the life of the mine. The effect of the smoothness f operation is particularly noticeable on a cable, a service of rom 10 to 14 months being obtained from each serving, while he strands will last for years.

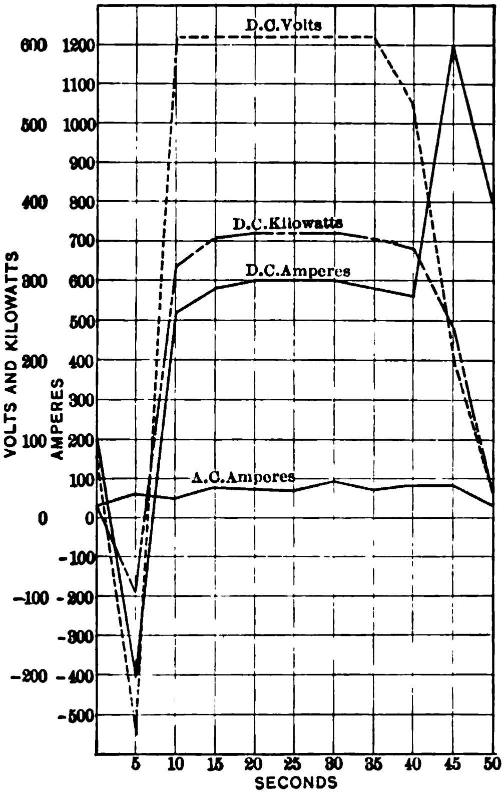


Fig. 7.—Five-Second Readings for Unbalanced Lowering to 1,200 Level Under Normal Conditions.

The control is so accurate that no chairs are needed in the haft to run on and off the waste-cars.

Should the brake be removed when the skip is empty, the peed with which it will descend unbalanced is remarkably low. This feature is clearly shown in Fig. 12, which is a set of readings taken at 1-min. intervals for a trip from the top to

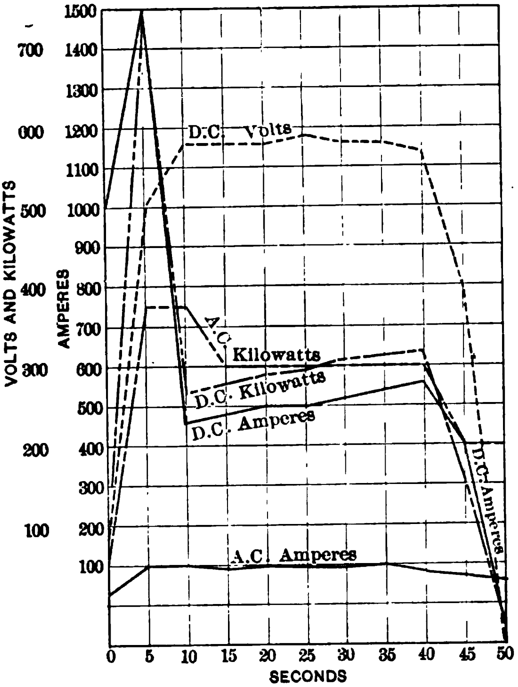


Fig. 8.—Five-Second Readings Taken During Balanced Hoisting Trip from 1,200 Level. Resistance-Switches Set to Operate at 310 kw. One-third Resistance in at End of Trip.

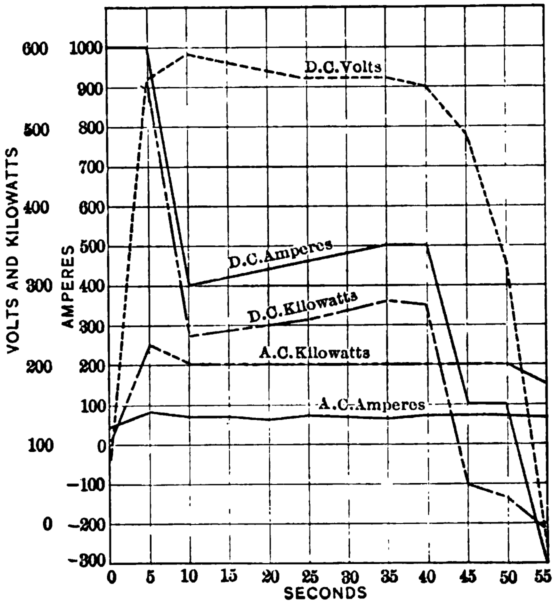


Fig. 9.—Five-Second Readings Taken During Balanced Hoisting from 1,200 Level. Resistance-Switches Set to Operate at 210 kw. Two-thirds Resistance in at End of Trip.

300 ft. No matter how slow the speed is, the pump-back action is taking place, and during the trips that waste and timber are being lowered into the mine, power is actually restored to the line.

The hoist requires but one man per shift to operate it. Another advantage of the hoist is its ability to operate for a short time, even though the power be accidentally interrupted. The running-lights in the hoist-room are all lighted from the exciter, which enables the operator to see as long as hoisting can be continued. Figs. 13 and 14 show 5-sec. readings for trips from the 900- and 1,200-ft. levels with a skip of ore after the A. C. power is cut off. During these tests, the speed of the

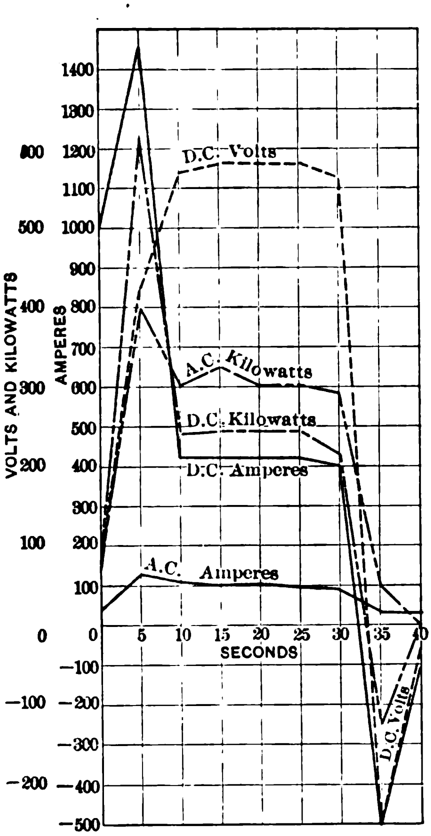


Fig. 10.—Five-Second Readings Taken During Balanced Hoisting Trip from 900 Level. Resistance-Switches Set to Operate at 310 kw. One-third Resistance in at End of Trip.

set became so low that the Tirril regulator short-circuited all the exciter-field rheostats, and its magnets failed to work. Without ore, as in handling men, the hoist is capable of making several trips after the A. C. power is off.

This installation has the disadvantage of consuming power, even though the hoist-motor is not in actual operation. This is more apparent than it would be if the hoist were operating from greater depths, or handling greater tonnage. The effect greater depth has on the efficiency is shown from the tests. From the 600-ft. level, the cost per 1,000 ft.-tons is

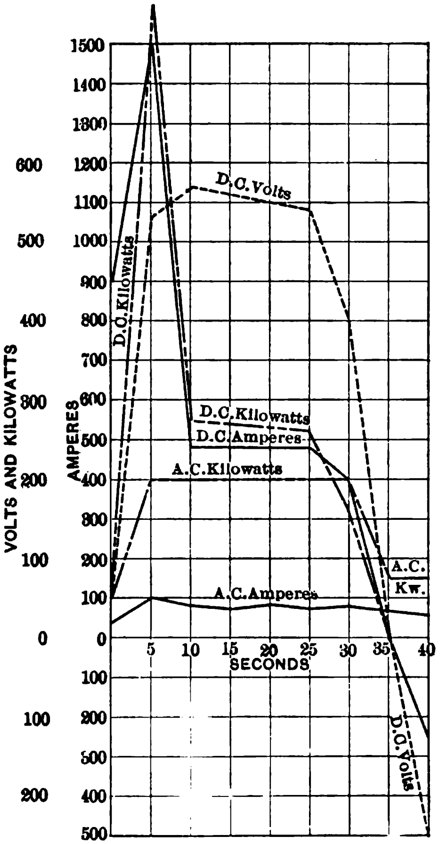


FIG. 11.—FIVE-SECOND READINGS TAKEN DURING BALANCED HOISTING FROM 900 LEVEL. RESISTANCE-SWITCHES SET TO OPERATE AT 210 KW. TWO-THIRDS RESISTANCE IN AT END OF TRIP.

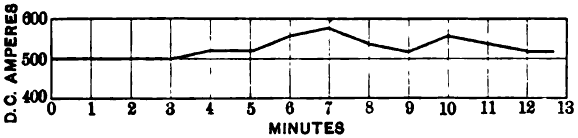


FIG. 12.—ONE-MINUTE READINGS TAKEN FOR UNBALANCED TRIP FROM TOP TO 300 LEVEL. THIS TRIP WAS STARTED AND STOPPED WITH THE BRAKE, THE CONTROLLER REMAINING IN THE OFF POSITION.

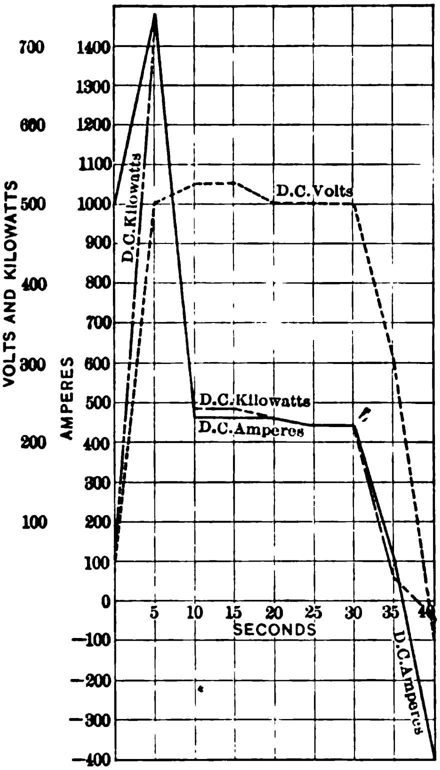


FIG. 13.—FIVE-SECOND READINGS TAKEN DURING BALANCED HOISTING FROM 900 LEVEL WITHOUT A. C. POWER, THE A. C. CIRCUIT-BREAKER BEING OPENED AT THE BEGINNING OF THE TRIP.

$0.014, from 900 ft. it is $0.0116, and from 1,200 ft. it is $0.0109. The effect greater tonnage would have on the cost per ton of output is shown by the following: Assuming that the mine double its output, the kw-hr. consumption per month would be increased by 17,231, at a cost of $132.50. The total cost for power for each ton of output would be lowered from $0.0313 to $0.0213.

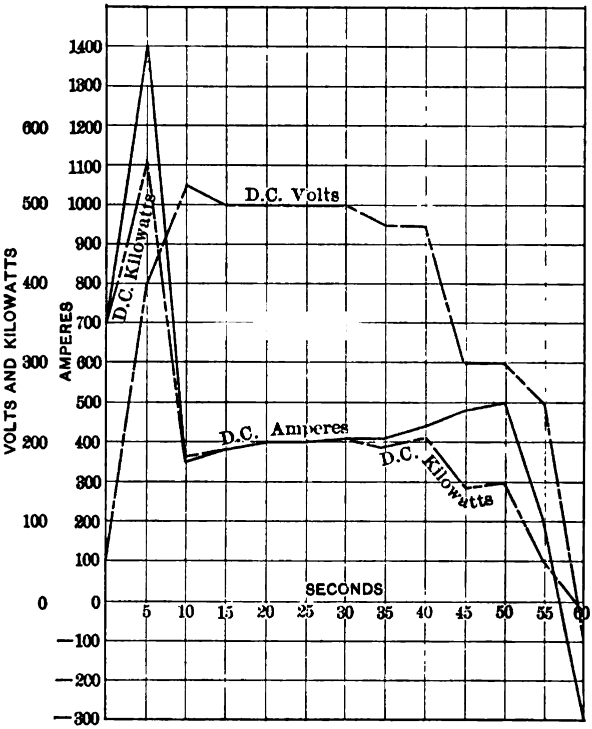


FIG. 14.—FIVE-SECOND READINGS TAKEN DURING BALANCED HOISTING FROM 1,200 LEVEL, WITHOUT A. C. POWER. THE A. C. CIRCUIT-BREAKER WAS OPENED AT THE BEGINNING OF THE TRIP.

Each hoisting problem has its own solution, and the Hecla Co. appears to have solved this one in a most satisfactory manner.

UBJECT TO REVISION.

[TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS.]

Efficiency-Engineering Applied to Mining.

BY GLENVILLE A. COLLINS, SPOKANE, WASH.

(Presented at a Meeting of the Spokane Local Section of the Institute, Feb. 17, 1912, and accepted for publication in the *Bulletin.*)

WHILE I am not at the present time engaged in active mine-management, my duties permit of extensive study and examination of many mines, mills, and reduction-works in various places throughout the United States.

Being much interested in efficiency-work in general, and an ardent reader of writings on this subject by Frederick W. Taylor and others, as applied to manufacturing-plants, and having served my apprenticeship from mucking to the engineering design of ore-dressing and underground plants, I hope that this paper will lead others to investigate the field more fully, to the improvement of general mine- and mill-operations, as well as the advancement, intellectually, physically, and financially, of the employee and his relations with the employer.

If a man who is prepared to look closely and observe broadly starts in Canada and zig-zags from camp to camp to southern Mexico, taking time casually to inspect mines, mills, and smelters, he will be pitifully impressed by the great difference in customs, and the general waste of labor, time, and money throughout the country. He will so seldom observe a dollar's worth of results given in exchange for a dollar spent that he will form a poor opinion of operating-efficiency and management. This highly-scientific department, requiring an amount of skill and executive ability equaled in few, if any, of the professions, certainly falls short of the standard established in our greater manufacturing industries.

It is easily said that this is due to the diversity of conditions in mining, milling, and smelting. But on second view it must be admitted that there is room for tremendous improvements in nearly every mining-organization. The proposal of such improvements is often met by the argument that any mine is but a temporary affair, yielding such profits as to make efficiency-work of no consequence; but any business which is profitable and worth doing at all is worth doing on a basis which gives the greatest return on the investment.

MANAGEMENT.

Neither an extremely dictatorial nor a lax management results in efficient operation in any mine; and harmonious relations between heads of departments are impossible under either of these two conditions. The mining business is such that every one connected with it is apt to be under strain from living-conditions, climate, and extreme duties of many characters, which makes it all the more necessary for the entire organization to be most harmonious.

Any scheme securing harmony of operations and increase of output at less expense to the owners, and still giving the employee better wages and living-conditions, is a benefit to the industry; and the higher development of the industry requires a combination of intelligence and labor, in which low cost of labor is to be desired by the employer and high wages by the workman. Diversity of mining-locations and conditions does not affect this truth.

It is, then, the purpose of this paper to advocate throughout the industry, high wages for workmen and low labor-costs to the employer, or what is generally known to manufacturers as the "Taylor System." I contend that this system is entirely feasible to the mining industry, through careful study and record of the time required to perform certain work.

It is often observed that the manager is a master in some branch of the business, and, as a rule, that branch is very well managed, while at the same time other departments under the same management are sorely neglected.

I would define good management as the accomplishment, expediently and cheaply, of exactly the result desired by the owners. When it is understood that a first-class workman can do from two to four times as much work in a given time as the average workman, the possibility of high wages for workmen with low labor-cost to the employer is appreciated.

It has often been noticed that if a workman is informed that it is possible for him, without undue fatigue, to produce from two to four times his normal work, he is incredulous, to say the least. Yet a really good workman, if given an opportunity to work at greater speed and earn from 20 to 100 per cent. more money, is only too glad to get the chance. But the exact percentage to be gained under this system by such terms of payment that a workman will do his best, can be determined only from carefully kept records, taken by a trained man, over long periods of time. It is not a matter to be decided off-hand, either by labor-unions or by directors.

The usual first-class workman will prefer to remain at his old system of work and wages rather than work at maximum speed unless the increase of pay is sufficient and permanent. For this reason it is not wise to make sudden or ill-advised changes, as the workmen then lose confidence in the system. It should never be referred to as a trial system, intimating the probability that it might not be a complete success. The thing to do is to be absolutely sure, then do it, regardless of the opinions of workmen. As soon as they begin to see the practicability of increasing their earnings and comforts, their opposition will cease, and they will become highly enthusiastic and thrifty.

As soon as the workman begins to realize that he is dealt with as an individual, with growing opportunity consistent with his energy and intellect, he leaves the ranks of organized shiftlessness and becomes an ambitious worker. This is also of great advantage to the employer, as a process of elimination which often develops extraordinary talents, otherwise locked up within a sullen breast.

The majority of manual laborers naturally make their task as easy as possible, eventually doing only that which they are forced to do. If an energetic workman is put with the rest on gang-work at day-wages, he gradually realizes that he is doing more work for the same pay, and in time slows down to the normal pace of the gang.

This may be exemplified by workmen tramming ore-cars. I have often seen them, on leaving the muck-pile at the face, mount the truck and take advantage of the usual down-grade and quick time to the dump, and then, after dumping the car, sit down on the pile from 5 to 20 min. for a rest, in order to give the muckers time to load easily the next car, and likewise "rest" before the trammers' return.

This form of systematic "soldiering" is often practiced to a remarkable degree undiscovered by managers.

When each operation and its required time for performance is absolutely determined and the workman put upon a just time-basis of minimum wages for a full shift's-work, with a bonus for extra work within the shift, then soldiering will have been effectually stopped.

As a further example, let us consider the case of drill-operators at work on ordinary drifting, using a machine-drill, but not doing their own mucking. Under the old system the drill-operator draws $4 per shift, and his helper $3. By careful time-study on this kind of work, it is found that, under old methods, a round of about 8 holes is drilled during the shift, with an average depth of 4 ft., making a total of 32 lin. ft. of hole per shift, at a labor-cost of $7, or an average of 22 cents per foot of hole drilled. This work is leisurely performed in the average drift in about 5 hr., after which from 1 to 2 hr. is used to clean out, load and fire the holes, and any balance of time left on the shift is wasted under the excuse of gases and bad air. In fact, there are very few managers who do not consider the above as a very fair day's-work, if they think of it at all.

After careful time-study we find that time is lost in securing proper tools for the job, their adjustment, delays caused by faulty conditions of equipment, and improper or no instructions from the foreman as to the work required, such as the most advantageous way of placing drill-holes. In trying to make it possible for the workman to earn from 20 to 100 per cent. more wages by using advanced methods to avoid loss of time, we find that it is very easy for the same men under the same conditions to put in the 32 lin. ft. of drill-holes, load and blast, in the first half of the shift, using the time required for ventilation in which to eat their lunch and look up any repairs or extras needed for the second half of the shift—the same work to be performed in the second half as in the first.

In the instance given, the driller and his helper are on a basis of $7 per shift and are not allowed to maintain their positions unless an average of 40 lin. ft. of hole is drilled, loaded and blasted per shift; but in excess of this footage a rate of 15 cents per linear foot of hole is paid, provided the excess is sufficient to average the hole-depths for efficient blasting.

This is equivalent to 51 per cent. increase in wages to the workmen and a reduction of 24 per cent. in the labor-cost to the employer; or, to be more explicit, when the crew makes 64 lin. ft. of hole per shift the driller receives $5.80 for his labor and the helper $4.80, while the employer pays $10.60 for labor in place of $14 under the old system.

It is of course necessary, in order to perfect such a system, that the tools be kept in good repair and proper means be provided to expedite mucking, tramming, and other portions of the work, in accordance with the general scheme.

To facilitate office-work in keeping records from which to compile pay-rolls and other data, it has been found economical and advantageous to use a card-system in which the workman makes out his own daily card. A form of card embodying this idea is shown in Fig. 1. The statement of the workman is easily checked by the foreman, who approves it before filing. All cards can be filed daily by dates without being indexed, and only handled once when making up the pay-roll or other work-reports. This file has many other advantages, among which are its accessibility, compactness, and simplicity. Again it serves as a "who's who" index of the best workmen, and requires but one office-man for a reasonably large company, covering mining as well as milling in all phases.

When required, the data on the workmen's cards can be plotted graphically so as to show, by inspection, operating-costs and results, without studying in detail each card. This is of great advantage in making annual reports to directors and stockholders, who usually dislike to study pages of detailed data, and prefer to inspect graphic charts which show at a glance the aggregates, and upon more careful examination the details, all given in a few sheets, instead of many pages of a lengthy and often poorly-written report.

It must be remembered that the average workman does not look far into the future, but demands immediate results for his labor. It is, therefore, very necessary to give workmen a just reward for their labor at the time of its performance, instead of holding out promises for the distant future. Hence it is wise, in establishing this system, to apply it very gradually, but firmly, and with definite data at hand.

On careful inquiry into the condition of workmen now working under this system in manufacturing-plants, it is found that their living-conditions improve greatly. It is almost impossible for one to drink to excess and keep up the work required. It is in reality a body of only first-class men who are all working together to secure large wages which are theirs irrespective of dull times and labor-strifes. The organization is in reality a labor-union in which the employer pays all expenses and gives the workman the best wages he can earn.

In adapting this system to mining one must expect to spend considerable money in order to secure proper fundamental data and to educate the men to the system. If these matters are given proper consideration, and the work is carried out gradually, step by step, the result will be thoroughly successful.

In mining, where it is desired to make the greatest possible output in the shortest time and at the least expense, it is evident that labor is the most important factor with which we have to deal. I believe that any experienced mine-manager will agree that a strong working-organization is of greatest value to the success of the industry. Mills and machinery which are destroyed by fire can easily be replaced with new capital; but it takes a long time to develop a strong working-organization.

I repeat that, in order to attain the most efficient management it is necessary to combine high wages with low labor-cost. The following four principles, as advocated by Frederick W. Taylor in his paper, Shop Management,[1] seem most simple of attainment:

1. *A Large Daily Task.*—Each man in the establishment, high or low, should have a certain clearly-defined task laid out before him. This task should not in the least degree be vague or indefinite, but should be circumscribed carefully and completely, and should not be easy to accomplish.

2. *Standard Conditions.*—Each man's task should call for a full day's-work, and at the same time the workman should be given such conditions and appliances as will enable him to accomplish his task with certainty.

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[1] *Transactions of the American Society of Mechanical Engineers*, vol. xxiv., pp. 1337 to 1456 (1903).

3. *High Wages for Success.*—He should be sure of large pay when he accomplishes his task.

4. *Loss in Case of Failure.*—When he fails he should be sure that sooner or later he will be the loser by it.

It is most advisable to organize what may be called a "planning"-department. I would advocate that this department consist of one or more specially-trained men for each branch of the industry. For example, one for the mine and one for the mill, the balance of the department to consist of the superintendents of the respective departments and the manager of the company—the superintendents and the manager not to take the initiative, but to work with the men who have had special training in each line.

By making use of a card-system in which the workman fills up his own card daily, to be approved by his foreman, it is often possible for the planning-department to handle all its office-work without extra help. The offices of this department should be situated as close to the work as possible, in order to keep in constant touch with its progress.

In starting work, each workman passes through the planning-department and secures his daily card, on the back of which the planning-department has designated plainly and exactly what his day's work is to be, and the tools he will require.

It hardly seems necessary to explain the advantages of such a department. It makes it unnecessary for the workman to think out his task or stop to hunt up tools, supplies, etc., as he is thus definitely instructed daily what to do, the tools to use and how to use them, leaving nothing for him to think of but performing his task in the most advantageous way and receiving his high wages. The planning-department naturally develops a standard of efficiency which if left to the foreman often suffers in all directions.

Mr. Taylor says in his paper that he has never been opposed by a strike, although he has been engaged for a greater part of his time, since 1883, in introducing this system of management in different parts of the country, and in a great variety of industries.

It certainly seems that a system which makes it possible for the workman to earn high wages should be advantageous in preventing labor-disturbances. This in itself would, in many

Number of Hole.	Depth.	Location. *No. 3 Drift.* REMARKS.	Number of Hole.	Depth.	Location. *No. 3 Drift.* REMARKS.
1	*4' 2''*				
2	*3' 11''*				
5	*4' 1''*				
6	*4' 0''*				
3	*4' 4''*				
4	*4' 0''*				
7	*3' 10''*	*Check.*			
8	*4' 3''*				
8	*32' 7''*	Total A. M.	*8*	*31' 2''*	P. M.

Name of Diller, *John Peters.* Helper, *Jack James.* Date, *Nov. 4–11.*

Drill, *2 1-2'' Holman.* *Peters, 5.80.*

Total Footage in Shift, *63' 9''*, *James, 4.80.*

Bonus 24' @ 15c. $3.45.

Approved,

Shift, *7–4.* *Joe. Hendy, Foreman.*

FIG. 1.—FORM OF TIME CARD.

Time in *7 A. M.*

Time out *4 P. M.*

Use *Holman 2 1-2'' No. 42.*

3–4 piece sets steel.

Column rig, No. 42.

Load Holes with 3 sticks Giant, using 5' fuse each charge. Tamp holes with clay.

DRILL HOLE DIAGRAM.

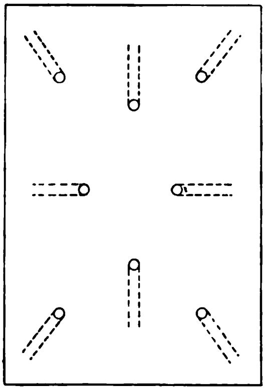

FIG. 2.—REVERSE SIDE OF TIME CARD.

cases, be a sufficient saving to the employer to warrant the adoption of the system, to say nothing of its possibilities in reducing operating-costs and increasing output.

This system is often regarded as applicable only to routine jobs; but it is so flexible that a daily task of several duties can be prescribed with decided success to one or more workmen, such as inspectors, janitors, millmen, yardmen, etc., whose work comprises many different functions.

Consider the usual work of a timekeeper, walking from place to place, and man to man, in all kinds of weather, spending only a few moments in each place, and trying to make an exact and detailed report on a separate card for each man. This could be done by each workman in a few minutes every day, thus giving the employer a most exact system of detailed work- and time-reports, while a timekeeper is only interested in the total time of work. A form of report card is shown in Figs. 1 and 2.

It should also be the function of the planning-department to analyze mechanical and other operations. In a large concentrator-plant, one cannot but notice the network of belting. It is usually next to impossible to drive concentrators in any other way than by belting; yet how many mine-superintendents and managers realize that an analysis of the tension on belting shows a vast difference in power, efficiency, life and usefulness of the belt. It has been found by exhaustive tests that belts properly cared for by a trained workman will average twice the pulling-power, and require much less care, than when handled according to the usual practice in the concentrators I have observed.

I have been especially impressed with the fact that the majority of large working mines have, as a rule, extremely efficient and up-to-date hoisting-apparatus, embodying not only the best safety-appliances, but operating economically; and yet in most of these high-class installations, when run by steam, there is a great disregard for the steam-plant design and equipment. Considering that the cost of any kind of steam-fuel is usually higher in a metal-mining district than almost anywhere else, we may well be surprised that there is less regard paid to its value in such places than in coal- and oil-camps where fuel is very cheap.

There are few mines, using more than a 100-h-p. boiler-plant, where it would not be possible to evaporate from one-fourth to two-thirds more water per pound of fuel burned than is now evaporated, and do it with less labor-expense.

Assuming $4 per ton as the cost of a fair grade of bituminous coal delivered on the grates of a boiler-plant of 500 h-p. rating, the saving of one-fourth of the fuel-expense means an annual saving of approximately $12,600. It is hardly possible to estimate the total value of such a saving (or the corresponding present waste) to the industry as a whole.

Space will not permit the consideration here of each phase of the mine- and mill-equipment; but what has been said of the possible value of improved apparatus and methods applies to nearly every branch of the industry and its relation to the results obtainable through a properly-organized efficiency-planning department.

In introducing this system, one cannot expect to attain the maximum output of the workmen at once; they gradually increase their working-capacity as they are taught efficient ways of handling themselves and their tools, and become accustomed to the new system.

In changing to this system I would advocate a start along the following lines:

1. The selection of a man of wide experience, not a miner, geologist, or mill-operator, but a man who has especial ability to observe what is actually going on, to take part in it, and to win the confidence and help of all the workmen.
2. The standardization of tools and methods throughout the mine, mill, and office.
3. The scientific study of time required to do unit-work in all lines.
4. The establishing of a card-system by which the workman can transmit the required information to the planning-department.
5. The making of a complete analysis of each piece of equipment used, and establishing a graphic-chart system of its operating-efficiency.

In making time-observations it is advisable to use a stop-watch, noting each observation and leaving nothing to memory. It is also necessary to record each detail of the operation, as a time-account of the total job is of little assistance to a first-rate planning-department.

In taking time-observations of machine-drill operation, I would advocate subdividing the time-observations into: (1) time required to place drill-column; (2) time to mount drill on column-arm; (3) time to make drill-connections ready for starting; (4) time to crank machine forward; (5) time to put down first bit; (6) time to withdraw bit; (7) time to place new bit; (8) time to crank to drilling-position; and so on through the operations of a shift.

It may seem unnecessary to go into such detail; but if a hundred observations of this character were taken, it would enable the planning-department to fix the exact time for many other complete jobs of a different nature and position. The observation of each step is more simple than when a combination of several steps is taken together. A large part of the work comprises certain operations of standard duration, such as cranking a drill forward or back, or removing its steel. These standards, once determined, have great latitude of application.

In a great many mines, mucking is often the cause of serious delay to drillers and general progress. This can be obviated by having the trammers help clear the space required for the drill-column, and letting the drillers stand on the pile while putting in the upper holes, the muckers in the meantime working from the outside of the pile. By the time the upper holes are in, the drillers can work to advantage on the lower holes without any footage-support, while the muckers finish clearing the pile.

In timing the mucking-operation, observations should be taken about as follows: Loading shovel-pan and straightening up; conveying to dumping-position and dumping; returning to loading-position; load on shovel in pounds. If we let x equal time in minutes for loading shovel and straightening up; y, the time for conveying to dumping-position and dumping; and z, the percentage of time per shift spent in resting, then the time in minutes required to shovel 1 ton equals

$$\left[(x+y)\frac{\text{Pounds in ton}}{\text{weight of shovelful in pounds}}\right](1+z).$$

Accurate analyses of the time-elements in this class of work make it possible to arrange economically the work in a mine so that mucking can keep pace with drilling- and other operations. Similar observations of time-elements will apply to a wide range of work usually classed under "miscellaneous jobs," in which no particular department-head has any special interest, and which, in general, is largely left to the handy man of the gang.

The principal objection which may be raised against the introduction of this system is the possibility of incurring the opposition of organized labor. But there are many considerations to the contrary. In the first place, we are to select first-class and intelligent workmen with whom to start the system, or, in other words, in introducing the system, one or two first-class men, carefully selected, are put on the new basis unknown to the other workmen. These men soon learn their earning-value, and have no desire to join a union of workmen below their grade.

To secure the greatest harmony between workmen and employer, the employer must be fair, fearless, and democratic. I do not believe that I have ever seen friction in any establishment in which the employer met his men half way and took the trouble to listen to their side of all matters. No manager ever lost any prestige through being well liked by his men.

By arranging a fixed plan through which each and every workman can in some degree express any private opinion he may have, labor-unions and strikes can be prevented. These suggestions and opinions are often of great value to the employer.

It is often possible for the management of larger concerns to organize a workman's club, which can meet periodically, and often with company officials, for written or oral discussions on subjects of interest to workmen. They usually appreciate lectures by trained men on improved methods of working, as well as an opportunity to read current literature or play clean games. While I do not believe in luxurious club-houses and conservatories, I do believe a well-organized workmen's club, provided with facilities for intellectual and bodily improvement, results in great contentment of workmen as a class. These, of course, should follow, and not precede, scientific management.

Such clubs replace entirely any desire on the part of a first-class workman to organize or join a labor-union. Let each employer organize in such a way as to make his men more contented, thereby, than with any outside inducements possible.

I realize that as long as an employer tries to drive his men, and at the same time pay them as poorly as possible, so long will there be demands for labor-unions or for other things of a rebellious nature.

The greatest success is to be obtained by leading workmen to better conditions; never by driving them. This does not mean that they are to have favors lavished upon them. It is only necessary that the employer have the whole interest of his workmen at heart—a spirit which is unconsciously radiated and felt by every one, and in exchange for which workmen will accomplish almost any task set before them.

It seems to me that the ultimate means of obtaining harmony of employer and employee is such a system of education as I have outlined, under which the workmen will expand their thoughts into actual thinking ability. While this would be a benefit to present workmen, I would go a step further and advocate the introduction into schools and colleges, and especially technical colleges, of a course of study dealing with advanced ideas on labor-problems and scientific management.

As I have already said, I believe the mining industry is greatly in need of more scientific organization and management; and I hope others will be led to investigate this subject more fully, with this end in view.

UBJECT TO REVISION.

[TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS.]

Mining-Methods at Nacozari, Sonora, Mexico.

BY D. C. LIVINGSTON, MOSCOW, IDAHO.

(Presented by invitation at a meeting of the Spokane Local Section of the Institute, Feb. 17, 1912.)

THE Pilares de Nacozari mine is located in Sonora, 75 miles outh of Douglas, Ariz. The town of Douglas is on the In ernational Boundary and is the place at which the ores from he Bisbee mines are smelted. The Moctezuma Copper Co. wns the mines at Nacozari, and the copper-concentrates hipped to the Copper Queen smelter, at Douglas, make a good melting-mixture with the Bisbee ores. The Moctezuma Cop er Co. and the Copper Queen smelter at Douglas, as well as he railroad from Douglas to Nacozari, are owned by the Phelps Dodge Co.

The ore-deposit occurs in the form of a large ellipse with a najor axis of approximately 2,000 ft. and a minor axis of bout 600 ft., the major axis bearing about 9° W. of N. The vhole of this ellipse is more or less mineralized; the surface eing principally an iron gossan with some occasional copper tains. Below the oxidized zone, which is not more than 50 t. deep, the minerals are pyrite and chalcopyrite, with occa ional seams of chalcocite, the latter being rare. There appears o be no well-marked zone of secondary enrichment, the oxi lized gossan over the greater part of the deposit changing uddenly to what is apparently the original unaltered ore. The opper-values are concentrated around the perimeter of the llipse, and it is principally around this perimeter that the nining is done. The ore mined averages a little more than 3 per cent. of copper, with a small amount of silver, less than n ounce per ton. Some ore of considerably higher grade han this has been shipped, however, but the above average is f the mill-run. The ore is wider near the two ends of the llipse than along the sides, and is mined in some cases to a vidth exceeding 200 ft. The country-rock for the first verti al 500 ft. is a volcanic acid breccia, probably rhyolitic; below this point it is more a basic breccia, probably andesitic. The contact is plainly marked on the surface by bluffs in one place, but it is almost impossible to trace it underground on account of the similar appearance of the two rocks when freshly broken. The change in formation has no apparent effect on the ore; if anything, it is somewhat richer but more bunchy in the lower formation, but there has not yet been enough work done to verify this statement. The contact between the mineralized rock of the ellipse and the unmineralized country-rock is plainly marked by a more or less definite wall, the rock on both sides of this wall, however, being the same. The ore is extracted from this wall towards the center of the ellipse until it becomes too low-grade to mine, although the whole of the interior of the ellipse would run almost 1 per cent. of copper.

This property presents a very interesting study of the origin of ore-deposits and was being examined and mapped last summer for the Moctezuma Copper Co. by J. M. Boutwell, formerly of the U. S. Geological Survey. There is a similar deposit at Mount Morgan, Australia, which also is in the form of an ellipse, though less than half the size of the one at Nacozari. I believe it is worked chiefly for the gold-values, and the whole of the ellipse is more or less workable. At the Braden copper-mine,[1] Chile, there is an ore-body of this same shape, but there the ore occurs along a contact of a volcanic neck which is intrusive in an older rock and different, while at Nacozari the country-rock and the interior of the ellipse are apparently the same.

The mine is opened by two 3-compartment shafts, situated one at each end of the ellipse, and an adit tunnel about a mile long which taps the deposit at a point about 600 ft. below the collar of the shafts. This adit is large enough to allow narrow-gauge, 25-ton railroad-cars to load from the mine-chutes, and it is chiefly due to this cheap handling of the ore that the enterprise has become so commercially successful. The railroad runs to the top of the bins above the concentrator, which is situated on the Nacozari railroad at the town of Nacozari, about 5 miles from the mine. From here the concentrates are shipped to the smelter at Douglas, Ariz. The daily output of ore is between 1,500 and 2,000 tons.

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[1] *Trans.*, xl., 743 to 746 (1909).

The levels in the mine are 100 ft. apart; and the general plan s to use every alternate level as a stoping-base, the inter- nediate levels being used for development, ventilation, and the listribution of waste to the stopes that are being filled. At the resent time practically all the ore is being mined above the dit-level, which is known as the "700," and at the present rate f output there should be, on a rough estimation, approximately n ore-supply of at least 10 or 12 years above this level. The haft at the north end has been sunk to the 1,000-ft. level, and ome work has been done on this level, as well as on the 800- nd 900-ft. levels.

Owing to the scarcity of timber in the neighborhood, and to he high price of imported timber, it has been necessary to dopt mining-methods which would eliminate the use of timber s much as possible. The nature of the rock is such that vhile it is not hard to drill or blast, it stands without timber n a most remarkable manner in most parts of the mine. This haracteristic of the rock has also prevented the using of the aving- and top-slicing methods, which are being used so suc- essfully at Cananea in some of the mines. Accordingly, some ystem of underhand stoping had to be devised, and the follow- ng two methods are used:

(1.) The old method, Fig. 1, which is room-and-pillar with vaste-filling as the work progresses.

(2.) The new method, Fig. 2, which is shrinkage with waste- lling after all of the ore has been extracted.

(1.) With the room-and-pillar method the mine-plan is laid ff into pillar-lines at right angles to the strike of the ore and laced 50 ft. apart, dividing it into alternate stopes and pillars. main working-drift is run approximately parallel to the strike, nd cross-cuts are turned off every 100 ft. in the pillars. From hese cross-cuts right-angle headings are turned off, and every 0 ft. run to the center of the stope and from these the sill- oor is cut out to the limits of the ore and to the pillar-lines, naking the stope 50 ft. wide. The roof, where necessary, is upported by props or cribs, and a 6-ft. cut is then taken upward vith stoping-drills, the blasted ore being drawn off immediately, nd filling run in from the level above through winzes to vithin 4 or 5 ft. of the roof. The short drifts from the cross- uts are timbered through the filling, and cribbed chutes and

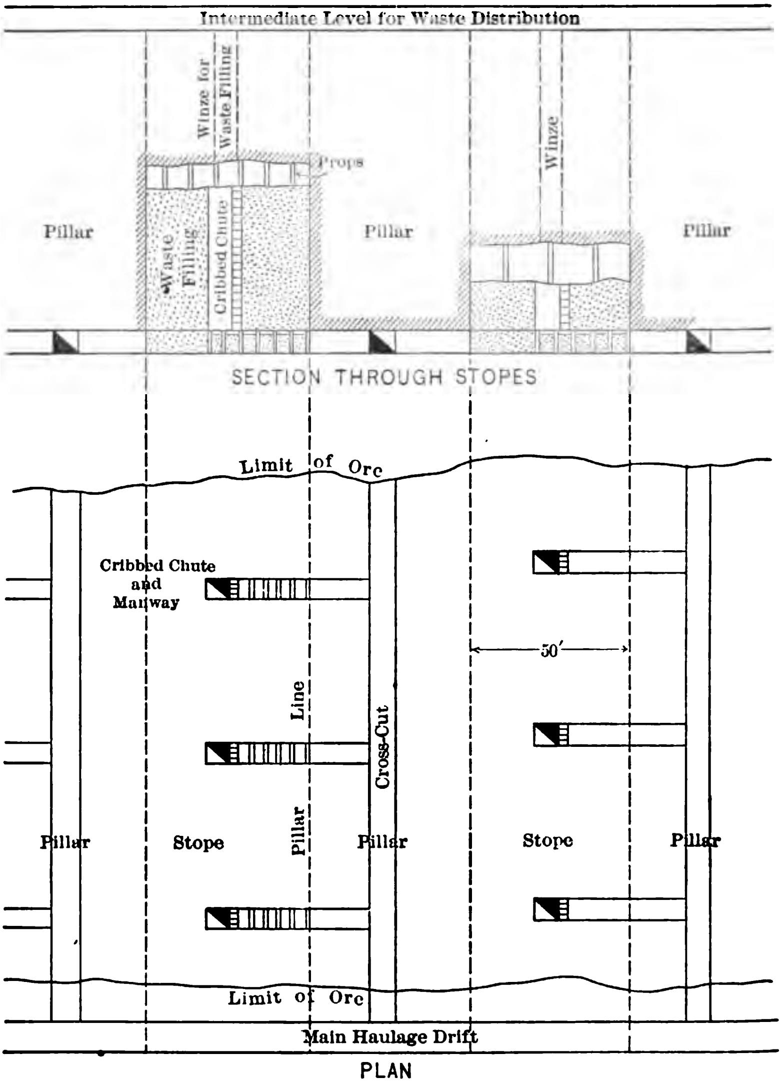

FIG. 1.—ROOM-AND-PILLAR SYSTEM OF MINING AS OPERATED AT THE PILARES DE NACOZARI MINE.

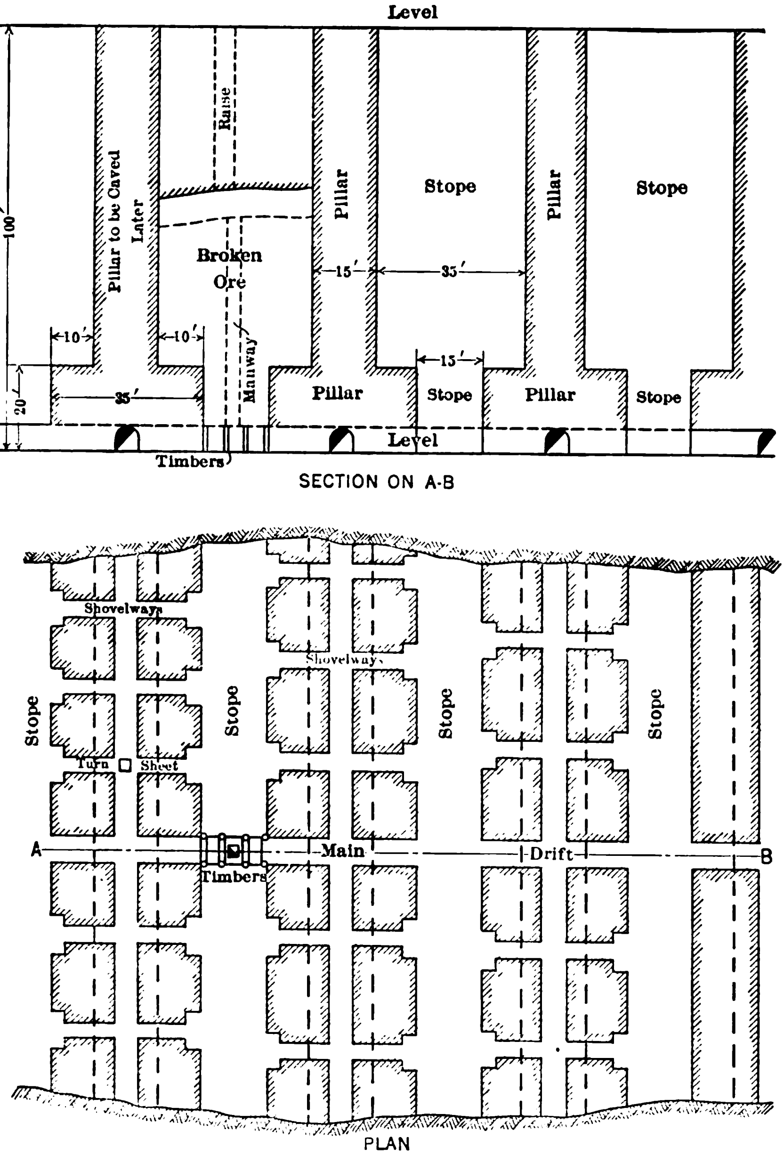

FIG. 2.—SHRINKAGE SYSTEM OF MINING AT THE PILARES DE NACOZARI MINE.

man-ways are carried up from the ends of these. In a stope 150 ft. long, there would be three of these chutes and man-ways.

There were no pillars being extracted when I was there, and it was probably the apparent difficulties of removing these pillars without losing a large amount of ore by mining with waste, which led to the adoption of the new method.

(2.) With the shrinkage method, designed by C. A. Smith, the use of timber is almost entirely eliminated, as not even timbered chutes are used. Cross-cuts are turned off from the main haulage-drift every 50 ft. parallel to the pillar-lines, and extending to the limits of the ore.

The next step is driving what are called the "stopes," which are breasts 15 ft. wide and the height of an ordinary drift. These are driven parallel to the cross-cuts, and midway between them, and are carried to the limit of the ore. From the cross-cuts "shovel-ways" are turned off at right angles at intervals of from 20 to 25 ft., and are driven to connect with the stopes. Turn-sheets are placed at the mouths of the shovel-ways. The ground is now ready to commence underhand stoping. The stopes are carried up the same width (15 ft.) with stoping-drills to a height of 20 ft. above the floor, and are then widened 10 ft. on each side making them 35 ft. in width, with 15-ft. pillars in between.

The machines are set up on the broken ore, enough being drawn out through the shovel-ways to keep a working-clearance. The stope is continued up in this manner to the next stoping-level. This work was started last summer, but it is evident that the main drifts will have to be timbered where the stopes cross them in order to keep the broken ore off of the track and cribbed man-ways built up through the broken ore at these points, but aside from this, with the exception of an occasional prop, no timber is used at all. There would be no absolute necessity for man-ways through the broken ore, as the stope can be entered by means of the winze from the level above.

The shovel-ways, which are merely rock-chutes, have been used in the mine for a number of years in the narrower portions of the vein instead of timber chutes. The reasons for their adoption are:

(1.) No timber required.

(2.) Ease and safety in blasting large boulders, which can be ıore safely and easily blasted in the shovel-ways than in the topes, and without interfering with the work.

(3.) All work, except timbering, is done by contract with he natives, and it was found that unless all the rock was roken in the stopes sufficiently small to pass a chute readily a difficult and expensive item in ground that breaks big where hrinkage is employed), it was just as cheap to have it shoveled nto the car as to draw off from a chute that was continually anging-up, and the expense of the timber was also eliminated.

In mining the pillars in the shrinkage system, I believe the lan is to put up raises in the pillars and from the raises to hatter the pillars with heavy shots, so that the pillars will cave hen the ore is drawn out of the stope. If this method works ut as expected at least 95 per cent. of the ore should be re-overed.

As regards the comparative costs of the two systems: stoping n the wide stopes by the shrinkage system had not progressed ufficiently in the summer of 1911 to give a complete compari-on, but what figures were available pointed to a considerable aving in the shrinkage over the room-and-pillar system. This hould be the case after the sill floor has been opened up, because, n the shrinkage system, there is no rock handled in the stopes ntil the ore is drawn out, whereas, in the room-and-pillar system oth the ore and waste filling have to be handled in the stopes n wheelbarrows.

Replaceable Lips for Elevator-Buckets.

BY H. J. MAGUIRE, WALLACE, IDAHO.

(Presented by invitation at a meeting of the Spokane Local Section, Feb. 17, 1912.)

THOSE familiar with mill-practice understand the work required of an average bucket-elevator, but I wish to call special attention to the wear on the buckets. I have been studying in what manner the life and service of the bucket might be improved, and I feel that an adjustable lip, attachable to each bucket without the aid of rivets, will be of material advantage.

For illustration, consider the feed-elevator of a typical mill in the Cœur d'Alène district. The ore is carried by water in a launder from the feed-rolls and dumped in the "boot" of the elevator, directly in front of the ascending bucket. Middlings from the return-rolls are similarly handled. Under this condition the bucket not only carries the ore, but also picks it up, and this latter requirement determines the life and service of the bucket. It might be said that it would be better practice to convey the feed to the elevator and dump it directly into the bucket, but at one mill at which this plan was tried the results were disappointing and the rubber belt was badly worn.

The wear of the bucket is confined to the lip and the corners which have to pick up the material. When the lip on a corner of a 7- by 12-in. malleable cast-iron bucket is worn down about 3 in., the bucket is of no further value as a carrier and has to be replaced. The worn bucket, containing about three-quarters by weight of the metal of the original bucket, has a scrap value of about 10 per cent. of that of a new bucket. To the cost of the bucket, however, should be added the time-cost of replacing the old bucket by a new one. This time-cost is considerable when the loss per hour, to cease operating the mill, is about $200.

It is out of the question to put lips on a bucket unless the latter was originally fitted with this device. For if it is attempted to fit lips upon buckets not originally provided with them, it will be necessary to drill holes for rivets and then spend considerable time fitting the lips to the irregular contours of the worn buckets. The improvement suggested is an adjustable attachment which can easily be fastened to, or removed from, the base of a bucket without the use of rivets. This attachment, to be made of a high-grade steel drop-forging or casting, will combine toughness, hardness, and tensile strength. The base of the bucket may be a cheap casting, possibly ordinary cast-iron. This combination of lip and base will not increase the cost of the improved bucket much, if any, above that of the ordinary bucket now in use, but will increase the life of the bucket, doing the same amount of work, about 400 per cent., with a corresponding decrease in time lost in changing buckets.

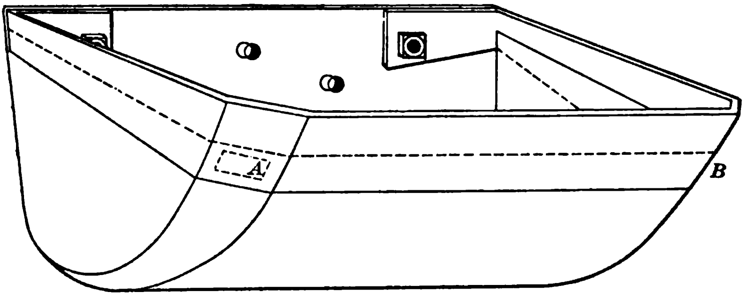

Fig. 1.—Isometric Projection of Bucket with Lip Attachment, Showing the Lip Bolted to Bucket.

This improved bucket is illustrated in Fig. 1, which shows a projection of the bucket with the attachment in place. Two bolts, one on each side of the elevator-belt, pass through the base and attachment and hold both fast to the belt. These end-bolts are 0.25 in. longer than the others. Fig. 2 shows the bucket with part of the lip removed, illustrating the manner in which it is fitted to the base and the opening at *A* which receives the lug *A*. There is another opening at *B*, which is not shown. Probably a more desirable arrangement than this would be to place the attachment inside the base, in which case the outside of the lip and the inside of the base would be rabbeted.

I am indebted to Stanley A. Easton, General Manager of the Bunker Hill & Sullivan mine, for the following data, which bear out my statement, and show that others have been trying to improve the service of elevator-buckets. He says that they use there 7.75- by 17-in. malleable cast-iron buckets, which give approximately 80 days' continuous service on middlings-elevators, in the old mill, handling 350 tons per day. Placing a manganese-steel lip on one bucket in a set of 72, added 35 days' service to the 80 days for the whole set. This set then gave a total service of 115 days. The cost of buckets per year, for 1,300 lin. ft. of belt handling 2,750 tons per day, is $1,436.80.

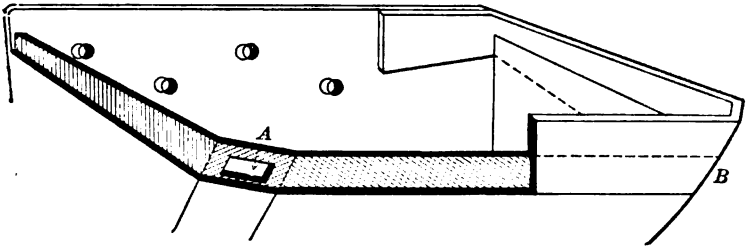

Fig. 2.—Bucket with Part of Lip Removed, Showing the Shoulder on which It Rests; also the Opening at *A* which Receives the Lug *A*, Fig. 1. Another Lug at *B*, Figs. 1 and 2, is not Shown in the Drawings.

On this basis, the total cost for elevator-buckets in the concentrators of the Cœur d'Alène district must be at least $10,000 per year. The saving by installing improved buckets can easily be estimated.

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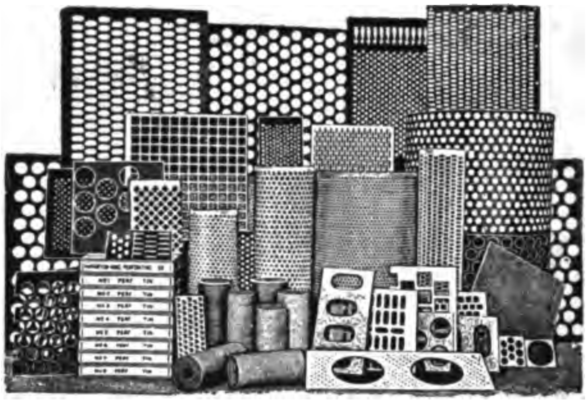

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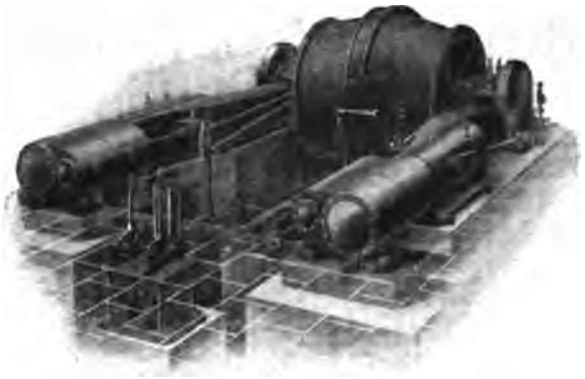

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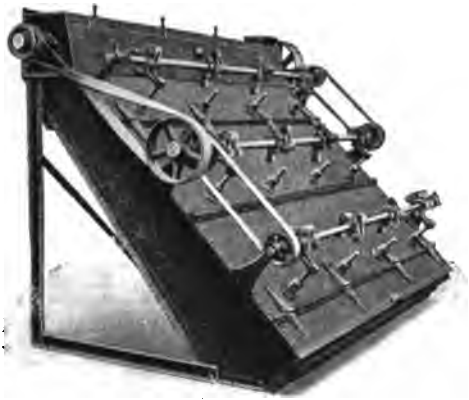

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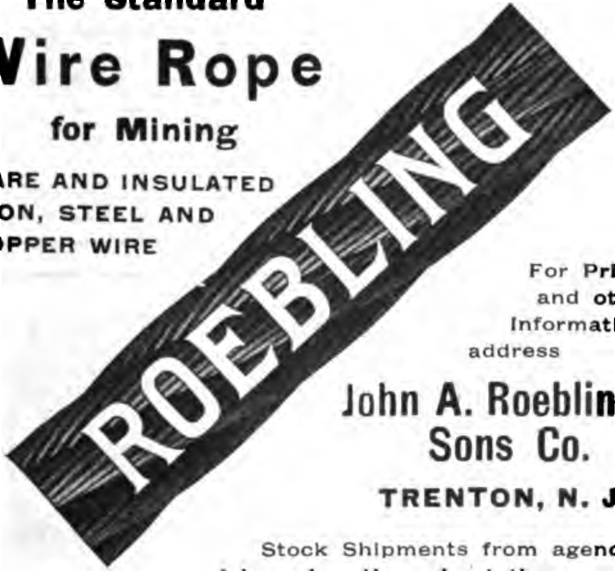

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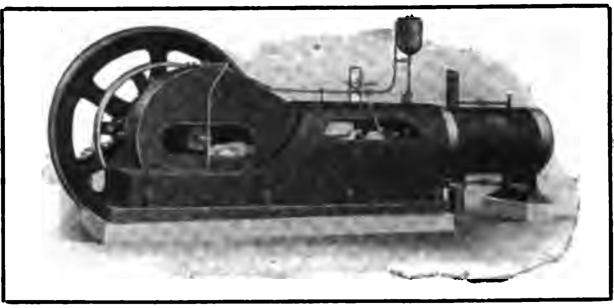

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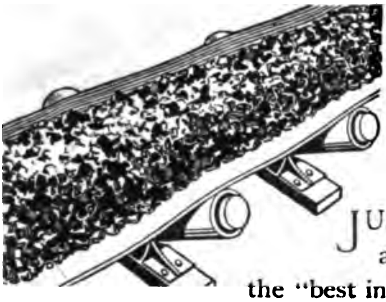

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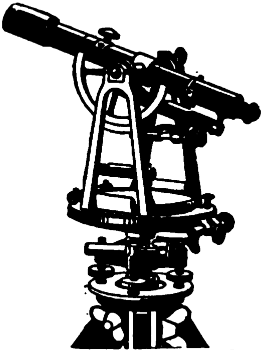

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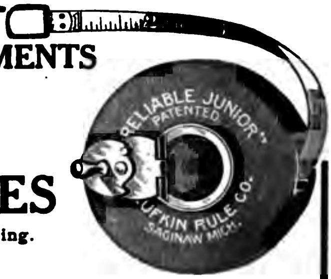
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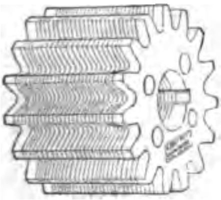

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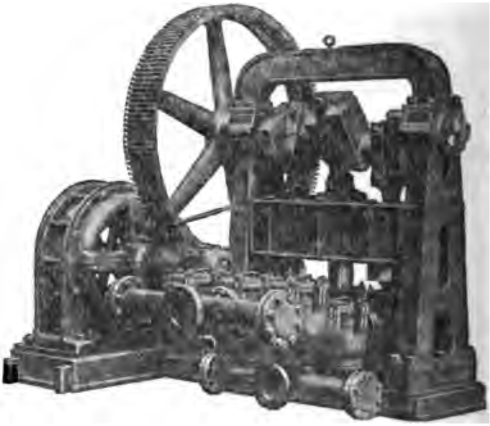

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No. 70 OCTOBER 1912

Bulletin of the American Institute of Mining Engineers.

CONTENTS:

SECTION I.—INSTITUTE ANNOUNCEMENTS.

PAGE

List of Officers and Committees, ... ii

Institute Announcements, ... v

Library, ... xvi

Membership, ... xxiii

SECTION II.—PAPERS AND DISCUSSIONS.

The Occurrence of Gold in the Eocene Deposits of Texas. By E. T. Dumble, ... 1021

Present Conditions of Mining in the District of Vladivostok, Siberia. By A. F. J. Bordeaux, ... 1025

The St. Helens Mining-District. By Horace V. Winchell, . . . 1037

A Titaniferous Iron-Ore Deposit in Boulder County, Colo. By E. P. Jennings, ... 1045

Clinton Iron-Ore Deposits in Kentucky and Tennessee. By S. Whinery, 1057

The Ta-yeh Iron-Ore Deposits, Hu-pei Province, China. By C. M. Weld, 1059

An Experience in the Use of Water-Power. By C. M. Myrick, . . 1071

The Sampling of Gold-Bullion. By F. P. Dewey, ... 1075

Centrifugal Machines for Ore-Grading and Ore-Concentrating. By Godfrey T. Vivian, ... 1103

Our National Resources and Our Federal Government. By R. W. Raymond, ... 1111

The Effect of Alumina in Blast-Furnace Slags. By J. E. Johnson, . 1123

Fires in Metalliferous Mines. By George J. Young, ... 1133

Recent Developments in Open-Hearth Steel-Practice. By N. E. Maccallum, ... 1153

Measurements and Relations of Hardness and Depth of Carbonization in Case-Hardened Steel. By Mark A. Ammon, ... 1167

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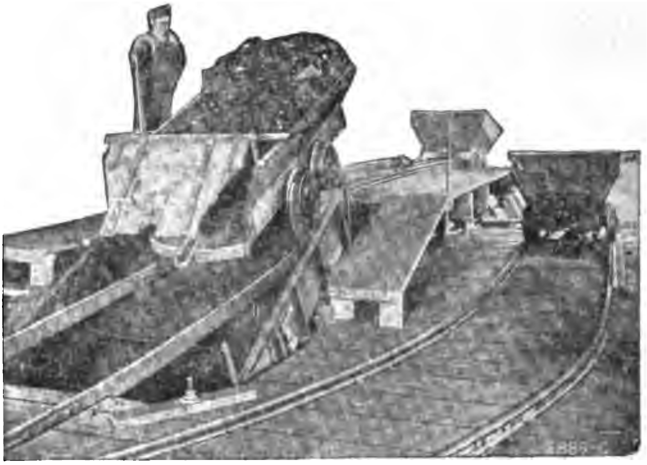

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TABLE OF CONTENTS.

SECTION I. INSTITUTE ANNOUNCEMENTS.

PAGE
List of Officers and Committees for the Year Ending February, 1913, . ii
Institute Announcements, ... v
Library, ... xvi
Membership, ... xxiii

SECTION II. TECHNICAL PAPERS.

[The American Institute of Mining Engineers does not assume responsibility for any statement of fact or opinion advanced in its papers or discussions.]

No. 1. The Occurrence of Gold in the Eocene Deposits of Texas. By E. T. DUMBLE, ... 1021
No. 2. Present Conditions of Mining in the District of Vladivostok, Siberia. By A. F. J BORDEAUX, ... 1025
No. 3. The St. Helens Mining-District. By HORACE V. WINCHELL, . 1037
No. 4. A Titaniferous Iron-Ore Deposit in Boulder County, Colo. By E. P. JENNINGS, ... 1045
No. 5. Clinton Iron-Ore Deposits in Kentucky and Tennessee. By S. WHINERY, ... 1057
No. 6. The Ta-yeh Iron-Ore Deposits, Hu-pei Province, China. By C. M. WELD, ... 1059
No. 7. An Experience in the Use of Water-Power. By C. M. MYRICK, . 1071
No. 8. The Sampling of Gold-Bullion. By F. P. DEWEY, ... 1075
No. 9. Centrifugal Machines for Ore-Grading and Ore-Concentrating. By GODFREY T. VIVIAN, ... 1103
No. 10. Our National Resources and Our Federal Government. By R. W. RAYMOND, ... 1111
No. 11. The Effect of Alumina in Blast-Furnace Slags. By J E JOHNSON, JR., 1123
No. 12. Fires in Metalliferous Mines. By GEORGE J. YOUNG, ... 1133
No. 13. Recent Developments in Open-Hearth Steel-Practice. By N. E. MACCALLUM, ... 1153
No. 14. Measurements and Relations of Hardness and Depth of Carbonization in Case Hardened Steel. By MARK A. AMMON, . . 1167

All communications concerning the contents of this Bulletin should be addressed to JOSEPH STRUTHERS, Ph.D., Secretary and Editor, 29 W. 39th St., New York, N. Y.

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For the year ending February, 1913.

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Any society of undergraduates at a technical school, comprising students in any branch of engineering, metallurgy, chemistry, geology, etc., may be recognized by the Council in its discretion as an Affiliated Student Society. A circular giving details of the plan of affiliation may be obtained on application to the office of the Secretary of the Institute.

The following societies have been placed by authority of the Council on the above list:

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INSTITUTE ANNOUNCEMENTS.

Special Notice.

The *Bulletin* is now entered at the Post Office at Second-Class Postage rate of one cent per pound, and in order to preserve this privilege it is necessary that the dues of members be paid within four months of January 1 of each year. If the dues are not paid within the period mentioned, a member's name must be removed from the regular subscription-list and the *Bulletin* mailed at the transient second-class postage rate of one cent for each four ounces or fraction thereof, prepaid by stamps affixed. It is therefore earnestly requested that dues be paid promptly—otherwise the Institute will be put to additional expense of postage and to added labor in removing and replacing names from the regular list, and in maintaining an additional separate mailing-list.

Nominating Committee.

At the meeting of the Council, June 28, 1912, the President announced the appointment of the following Nominating Committee to present names for officers of the Institute to be voted upon at the Annual Business Meeting, Feb. 18, 1913.

Philip N. Moore, St. Louis, Mo., *Chairman.*
John Birkinbine, Philadelphia, Pa.
David W. Brunton, Denver, Colo.
Seeley W. Mudd, Los Angeles, Cal.
Charles P. Perin, New York, N. Y.
Arthur L. Walker, New York, N. Y.

A New Volume on Ore-Deposits.

Ore-Deposits—a continuation of the "Posepny" Volume.

Comprising Papers Descriptive of Ore-Deposits and Discussions of Their Origin, Edited, with an Introduction, by Dr. S. F. Emmons. The volume contains also a Biographical Notice of Dr. Emmons by his associate and friend, Dr. George F. Becker, and a comprehensive Bibliographical Index of the Science of Ore-Deposits, prepared by Prof. John D. Irving, of the Sheffield Scientific School of Yale University. Dr. Emmons had finished his editorial work and written his Introduction before his lamented death in 1910; and the Volume contains his last words upon the subject to which he had given the work of his life, and on which he was justly regarded as the foremost authority.

Further details concerning time of publication, price, etc., will follow.

Adjourned Annual Business Meeting of the Institute.

The Adjourned Annual Business Meeting of the Institute, called for Oct. 7, 1912, was duly held at Institute Headquarters, New York City, President Kemp presiding.

The amendments to the Constitution, which were presented for acceptance or rejection at the convening of the Annual Business Meeting, Feb. 20, 1912, and carried over by vote for future action, were withdrawn by the Chairman of the Committee of the Board of Directors and the Council.

In accordance with the Constitutional requirements, certain proposed amendments to the present Constitution and By-Laws of the Institute were submitted in writing by Messrs. C. R. Corning and George C. Stone. Certain proposed amendments to the Constitution were also presented by Prof. Joseph W. Richards in behalf of a Special Committee appointed by the Board of Directors and the Council.

It was voted to print all of the proposed amendments and send copies to the membership at large for information and study, with the view of giving, especially to those residing in the Middle West, an opportunity for an oral or written discussion at the coming meeting in Cleveland, Oct. 28 to 31, 1912. These amendments will be presented in final form at an adjournment of the present adjourned Annual Business Meeting, which was voted to be called for Tuesday, Nov. 12, 1912, at the Institute Headquarters in New York City.

A communication was received from members in San Francisco and one from those in Los Angeles, making certain recommendations. These communications will be duly printed and sent to the membership with the proposed amendments.

It was voted to accept the report of the Committee of Five, and to discharge the Committee with thanks.

It was voted to order the report, with its accompanying documents, to be placed on file in the archives of the Institute.

It was voted to appoint a Special Committee of Three to draft a plan for substituting one body of management for the present dual management through the Board of Directors and the Council. Prof. Joseph W. Richards, Charles Kirchhoff, and Charles F. Rand were subsequently appointed on this Committee.

The meeting adjourned to reconvene on Tuesday, Nov. 12, 1912, at 10.00 a.m. in the Institute Headquarters, New York City.

Cleveland Meeting.

As already published in Announcement, No. 7, mailed to the membership under date of Sept. 27, 1912, the 103d meeting of the Institute, for the reading and discussion of professional papers, will be held in Cleveland, Ohio, beginning Monday evening, October 28, 1912.

Institute Headquarters will be established at the recently completed Statler Hotel, where a bureau of information in charge of

Mr. C. B. Murray, Secretary of the Local Committee, will be maintained. The early sessions will be held in one of the rooms of this hotel, and the concluding session in the Electricity Building of the Case School of Applied Science.

The arrangements for the meeting, apart from the technical sessions, are in the hands of the Local Committee, Mr. D. T. Croxton, *Chairman;* Mr. C. B. Murray, *Secretary,* Perry-Payne Building, Cleveland, Ohio, to whom all inquiries concerning local matters, except those referring to papers and discussions for the technical sessions, should be addressed. The control of the papers and discussions will be exercised by the President and Secretary of the Council as usual.

In order to facilitate the completion of the preliminary arrangements for this meeting, members who have not yet notified the Secretary's office of intention to be present at this meeting, are earnestly requested to do so as soon as possible, stating intention to be present, and giving the names of accompanying guests.

Hotel Accommodations.

Hotel rates are:

Statler Hotel, European plan, with shower bath, $2 per day and up; with tub bath, $2.50 per day and up. Extra person, $1.50 additional. Rooms may be secured in advance by correspondence direct with the hotel manager, Mr. James P. A. O'Connor, Statler Hotel, Cleveland, Ohio.

Hollenden Hotel, European plan, $2 per day and up; with bath, $2.50 per day and up.

Colonial Hotel, American plan, without bath, $3.50 per day and up; with bath, $4 per day and up. European plan, without bath, $2 per day and up; with bath, $2.50 per day and up.

In engaging rooms, members or guests should mention their connection with the Institute.

Train Service.

No special reduction of railroad fares for attendance at this meeting is practicable, and members will make their own arrangements in this respect.

Committees:

Executive: D. T. Croxton, *Chairman;* C. B. Murray, *Secretary;* Wm. G. Mather, *Treasurer;* F. B. Richards, A. W. Smith, S. T. Wellman.

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Program.

Details of the technical sessions of the meeting, and visits to points of engineering and scenic interest, will be given in the program of the Local Committee, which will be furnished, together with an identification badge, to every member or guest upon registration at Headquarters in the Statler Hotel.

Members wishing to visit during the week any individual plant or work in Cleveland or vicinity will be assisted to do so by means of suitable introductions and guidance upon notice given by them in advance to Mr. C. B. Murray.

The following provisional program has been arranged, subject to possible but not probable change by the Local Committee:

Monday, October 28, 1912.

Afternoon: Registration at Headquarters, Statler Hotel.
8.15 p.m.—Informal Social Meeting at Headquarters.

Tuesday, October 29, 1912.

10.00 a.m.—Technical Session, 2.00 p.m.—Technical Session, Evening.—Informal Smoker and Popular Addresses,	Hotel Statler.

Wednesday, October 30, 1912.

10.00 a.m.—Technical Session at Statler Hotel.
Afternoon.—Trip.
7.30 p.m.—Subscription Banquet, members and guests, including ladies.

Thursday, October 31, 1912.

10.00 a.m.—Technical Session at Case School of Applied Science.
Afternoon.—Trip.
Evening.—Unassigned.

Papers Announced.

While some of the papers on the following list will necessarily be "read by title" only, it is hoped that most of them will be presented by their authors. However, for the purpose of discussion, in the absence of the author, any paper may be called for presentation in oral abstract.

Printed copies of many of these papers will be on hand at the sessions. Authors intending to be present and desiring to have time assigned to them are requested to notify Dr. Struthers, in writing, immediately. In the absence of such prompt notice, the desired assignment of time cannot be definitely promised.

Members desiring to see any of these papers for the purpose of discussion at the meeting, are requested to communicate with Dr. Struthers, who will forward in reply advance copies of the desired papers, if they are in print, or will give opportunity during the meeting for an inspection of the manuscripts.

IRON AND STEEL DIVISION.

Method of Surveying and Sampling Diamond-Drill Holes, by E. E. White, Ishpeming, Mich.

Cuyuna Iron-Ore Range, by Walter A. Barrows, Jr., Duluth, Minn., and Carl Zapffe, Brainerd, Minn.

The Manufacture of Coke, by W. H. Blauvelt, Syracuse, N. Y.

The Manufacture of Coke, by F. E. Lucas, Sydney, C. B.

Notes on Ruff's Carbon-Iron Equilibrium Diagram, by Henry M. Howe, New York, N. Y.

The Utility of Efficiency-Records in the Manufacture of Iron, by John J. Porter, Staunton, Va.

The Fuel-Efficiency of the Cupola-Furnace, by John J. Porter, Staunton, Va.

Blowing-In of Blast-Furnaces, by Ralph H. Sweetser, Columbus, O.

On Blast-Furnace Air-Pressures, by J. E. Johnson, Jr., Ashland, Wis.

Notes on Titanium, and the Cleansing Effect of Titanium on Cast-Iron, by Bradley Stoughton, New York, N. Y.

N. M. Wittorff's Preliminary Investigation of Primary Crystallization and Subsequent Physico-Chemical Permutations in Iron-Carbon Alloys Containing More than 4 Per Cent. of Carbon, by Bradley Stoughton and Simon Marcus, New York, N. Y.

Microstructure of Iron and Steel, by William Campbell, New York, N. Y.

*Recent Developments in Open-Hearth Steel Practice, by N. E. Maccallum, Phœnixville, Pa.

The Heat-Treatment of Steel-Castings, by C. D. Young, Rolla, Mo.

Note on the Case-Hardening of Special Steels, by Albert Sauveur and G. A. Reinhardt, Cambridge, Mass.

The Action of Various Commercial Carbonizing Materials, by Robert R. Abbott, Cleveland, Ohio.

*Measurements and Relations of Hardness and Depth of Carbonization in Case-Hardened Steel, by Mark A. Ammon, Cleveland, Ohio.

*The Effect of Alumina in Blast-Furnace Slags, by J. E. Johnson, Jr., Ashland, Wis.

Quality of Charcoal Iron, by J. E. Johnson, Jr., Ashland, Wis.

*A Titaniferous Iron-Ore Deposit in Boulder County, Colo., by E. P. Jennings, Salt Lake City, Utah.

*The Ta-yeh Iron-Ore Deposits, Hu-pei Province, China, by C. M. Weld, Low Moor, Va.

*Clinton Iron-Ore Deposits in Kentucky and Tennessee, by S. Whinery, New York, N. Y.

Concentration of Iron-Ores, by N. V. Hansell, New York, N. Y.

Recent Developments in the Inspection of Steel Rails, by Robert W. Hunt, Chicago, Ill.

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* Published in this number of the *Bulletin*.

†Electric Heating and Removal of Phosphorus from Iron, by A. E. Greene, Chicago, Ill.

†The Function of Slag in Electric Steel-Refining, by R. Amberg, Pittsburg, Pa.

†The Fuel-Efficiency of the Cupola-Furnace, by John J. Porter, Staunton, Va.

Melting Iron in the Cupola-Furnace, by R. Moldenke, Watchung, N. J.

GENERAL SUBJECTS.

*Centrifugal Machines for Ore Grading and Ore-Concentrating, by Godfrey T. Vivian, Cornwall, England.

*The St. Helens Mining District, Skamania, Wash., by Horace V. Winchell, Minneapolis, Minn.

*Mine-Fires in Metalliferous Mines, by George J. Young, Reno, Nev.

*Present Conditions of Mining in the District of Vladivostok, Siberia, by Albert F. J. Bordeaux, Vladivostok, Siberia.

*The Sampling of Gold-Bullion, by Frederick P. Dewey, Washington, D. C.

*An Experience in the Use of Water-Power, by C. M. Myrick, San Francisco, Cal.

*Our National Resources and Our Federal Government, by R. W. Raymond, New York, N. Y.

The Wood Flotation Process, by Henry E. Wood, Denver, Colo.

*The Occurrence of Gold in the Eocene Deposits of Texas, by E. T. Dumble, Houston, Tex.

A Graphic Solution of D'Arcy's Formula for the Transmission of Compressed Air in Pipes, by Nathaniel Herz, Lead, S. D.

The Alundun Extraction-Thimble Used in the Determination of Copper, by L. W. Bahney, New Haven, Conn.

Notes on the Metallography of Alloys, by William Campbell, New York, N. Y.

Constitution and Melting-Points of a Series of Copper-Slags, by Charles H. Fulton, Cleveland, Ohio.

The Ultimate Source of Ores, by Blamey Stevens, Temascaltapec, Mexico.

Discussion of the paper of F. G. Cottrell, Fume-Precipitation, by Richard Peters, Jr., Barker, Delaware Co., Pa.

†Notes on Bag-Filtration Plants, by Anton Eilers, New York, N. Y.

†Development of the Parkes Process in the United States, by Ernst F. Eurich, New York, N. Y.

†Sulphatizing-Roasting of Copper-Ores and Concentrates, by Utley Wedge, Philadelphia, Pa.

†Development of the American Water-Jacket Lead Blast-Furnace, by R. C. Canby, Wallingford, Conn.

†Alloys of Cobalt with Chromium and Other Metals, by Elwood Haynes.

†Heat-Losses in Furnaces, by F. A. J. Fitzgerald, Niagara Falls, N. Y.

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* Published in this number of the *Bulletin*.

† Presented at Joint Meeting with Eighth International Congress of Applied Chemistry, September, 1912.

Iron and Steel Division.

COMMITTEE.

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Charles K. Leith,	Albert Sauveur,	William R. Webster.

A complete list of the iron and steel papers to be presented at the coming meeting of the Institute will be found in the announcement, printed elsewhere, of the program of the Cleveland meeting.

It includes papers by E. P. Jennings, on a Titaniferous Iron-Ore Deposit in Boulder County, Colo.; by C. M. Weld, on the Ta-yeh Deposits in China, and by S. Whinery, on Clinton Deposits in Kentucky and Tennessee.

Exploration is dealt with in E. E. White's paper on the Method of Surveying and Sampling Diamond-Drill Holes, while the preparation of ores is represented by N. V. Hansell in the Concentration of Iron-Ores.

W. H. Blauvelt and F. E. Lucas will present papers on the Manufacture of Coke. Papers relating to Blast-Furnace Practice are those of J. E. Johnson, Jr., on the Effect of Alumina in Blast-Furnace Slags, and on Charcoal-Iron; of Ralph H. Sweetser, on the Blowing-In of Blast-Furnaces, and of John J. Porter, on the Utility of Efficiency-Records.

Cupola-practice is represented by J. J. Porter, on Fuel-Efficiency of the Cupola; and by R. Moldenke, on Melting Iron in the Cupola-Furnace, while Bradley Stoughton's Notes on Titanium and the Cleansing-Effect of Titanium summarizes the literature on the subject and records the results of many special experiments.

Electric Heating and Removal of Phosphorus from Iron is the title of a paper by A. E. Greene,

Recent Developments in Open-Hearth Practice, by N. E. Maccallum; and the Function of Slag in Electric Steel-Refining, by R. Amberg, enter the field of the manufacture of steel; while Robert W. Hunt's Recent Developments in the Inspection of Steel Rails deals with one of the most important finished products.

Developments in Metallography are brought forward by a weighty paper by H. M. Howe on Notes on Ruff's Carbon-Iron Equilibrium Diagram; by Albert Sauveur's and C. A. Reinhardt's Case-Hardening of Special Steels; and by William Campbell's Notes on the Microstructure of Steel. The latter applies particularly to special steels, other phases of which are dealt with by Robert R. Abbott in his Comparison of the Action of Various Carbonizing Materials; and by Mark A. Ammon in his Measurements and Relations of Hardness and Depth of Carbonization in Case-Hardened Steel.

The program is therefore a varied and comprehensive one, and will offer exceptional opportunities for interesting and instructive meetings and an animated discussion.

New York Local Section.

Executive Committee.

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E. GYBBON SPILSBURY, *Vice-Chairman.*
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THOMAS ROBINS.
LOUIS D. HUNTOON, *Secretary-Treasurer,*
165 Broadway, New York.

The first annual meeting of the New York Local Section of the American Institute of Mining Engineers was held Sept. 19, 1912, in the Engineering Societies Building, 29 West 39th Street, New York, N. Y.

Colonel William M. Black, of the Corps of Engineers, U. S. Army, gave an illustrated talk, entitled, "The Story of the Maine," from the time it entered Havana harbor to the time the wreck was buried at sea. The lecture covered the events immediately preceding and following the destruction of the vessel, the determination of the United States government to remove the wreck from the harbor, the formation of the project, and the engineering features connected with the removal. A most interesting description was given of the construction of the coffer-dam, elliptical in shape, built of steel cylinders. These cylinders were made of narrow interlocking plates of steel in two sections, which were driven into the mud with a pile-driver. To draw the ends of the cylinders together, the last 15 plates were driven simultaneously. After the cylinders were in place, wings, consisting of interlocking sheet-piling, and interlocking with the cylinders, were sunk on the outside. The cylinders and the space between them and the wings were then filled with clay, making the coffer-dam water-tight. Colonel Black told of the great care exercised by the officers in making a close examination as the unwatering proceeded and the wreck came into view, and the taking of detailed notes for the report to be submitted to the government.

The description of other engineering details, such as cutting portions of the wreck loose with acetylene blow-pipes, and loosening the wreck from the mud by pumping water below it, made the lecture most profitable and enjoyable.

After the meeting the members adjourned to the offices of the Institute, where refreshments were served. The attendance at the meeting was 150, and the lecture was thoroughly appreciated by all.

Arrangements have been made for other lectures, followed by a social hour in the offices of the Institute, where the members will have an opportunity of meeting one another.

The Executive Committee desire to interest all members in the meetings and will appreciate suggestions.

LOUIS D. HUNTOON, *Secretary-Treasurer,*
165 Broadway, New York, N. Y.

Joint Meeting of the Institute with Eighth International Congress of Applied Chemistry.

The Institute joined forces with the recent International Congress by holding several joint sessions with Section III, Mining and Metallurgy, of which Dr. Joseph W. Richards was president, J. B. F. Herreshoff, vice-president, and many other members of the Institute active participants.

On Sept. 6, joint sessions were held in Schermerhorn Hall, Columbia University, which were numerously attended, and at which the following papers were read and discussed:

R. Adam. Micrographie des fontes speciales de fer.
* Bradley Stoughton. The Influence of Titanium on the Strength of Iron Castings.
† N. S. Maccallum. Recent Developments in Open-Hearth Steel-Practice.
A. Ledoff. Iron Coke.
* E. P. Mathewson. Development of the Reverberatory Furnace for Smelting Copper Ores.
* F. Laist. Chemistry of the Reduction Process in Use at Anaconda, Montana.
* Utley Wedge. Roasting Copper-Ores and Concentrates Preliminary to Leaching for Copper-Values.
‡ H. O. Hofman and M. Wanjukow. The Decomposition of Metallic Sulphates at High Temperatures in a Current of Dry Air.
*R. C. Canby. Development of the American Water-Jacket Lead Blast-Furnace.
F. Cazin. The Evolution of the Modern Lead Blast-Furnace.
F. Guiterman. Silver-Lead Smelting.
* E. F. Eurich. Zinc-Desilverization.
* A. Eilers. Notes on Bag-Filtration Plants.
A. S. Dwight. The Dwight-Lloyd Process of Roasting and Sintering.
W. M. Corse. Modern Manganese-Bronze Tests.
* Elwood Haynes. The Alloys of Cobalt with Chromium and Other Metals.
* J. M. Camp. The Method of the United States Steel Corporation for the Commercial Sampling and Analysis of Pig-Iron.

On Sept. 7, a similar joint session was held, with the addition of the Section Xa, Electrochemistry, of the Congress and members of the American Electrochemistry Society. The topic of the session was "Electric Furnaces," and the following papers were read and actively discussed:

* F. A. J. Fitzgerald. Heat-Losses in Furnaces.
C. H. Vom Baur. Electric Induction and Resistance Furnaces.
* A. E. Greene. Electric Heating and the Removal of Phosphorus from Iron.
* R. Amberg. The Function of Slag in Electric Steel-Refining.
A. Stansfield, D. L. McLeod, and J. W. McMahon. The Electrical Resistivity of Fire-Bricks at High Temperatures.
S. A. Tucker and H. L. Read. Fixation of Nitrogen by Alumina and Carbon.
W. N. Lacy. On the Electrolytic Refining of Silver-Bismuth Alloys.

This quadruple joint-meeting was held in the large chemical lecture-room of Havemeyer Hall, Columbia University, with an attendance so large that "standing room only" was available for late comers. The discussions were lively, and the presence of many well-known metallurgists and electric-furnace engineers from this country and abroad gave real international flavor to a very successful meeting.

The International Congress, in general session, passed an official vote of thanks to the Institute for its co-operation.

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* To be published later in the *Bulletin* of the Institute.
† Published in this number of the *Bulletin*.
‡ *Bulletin* No. 69, Sept., 1912, pp. 889 to 943.

Library Research-Work.

The attention of members of the Institute is again directed to the research-work done by the librarian and his assistants, which should attract special attention from those members who have no access to the literature of subjects in which they may be interested.

During the year 1911 there were 143 searches made for members and non-members of the Founder Societies, and copies of the references have been preserved for the use of others. This work has been largely based on requests sent in by mail, from Japan, South Africa, Mexico, Canada, and England, as well as from different parts of the United States. The Librarian is confident that if it were more widely known that the library is equipped to undertake researches, the demand would increase beyond the ability of the present force to handle it. The library receives more than 700 technical periodicals which are available through the indexes for this special purpose.

Back Volumes of the Transactions.

The Board of Directors has authorized the following offers of sets of back volumes of the *Transactions*, at considerably reduced prices, to Members, Libraries, and Scientific Societies:

		Per Set.
I.	Five volumes, bound in half-morocco, from No. 36 (1906) to No. 40 (1910), ...	$20
II.	Ten volumes, bound in half-morocco, from No. 31 (1902) to No. 40 (1910), including Mexican Volume, . . .	35
III.	Twenty volumes, bound in half-morocco, from No. 21 (1893) to No. 40 (1910), ...	50
IV.	Thirty volumes, bound in half-morocco, from No. 11 (1883) to No. 40 (1910), ...	60
V.	Thirty-nine volumes, bound in half-morocco, from No. 1 (1873) to No. 40 (1910), with the exception of No. 10 (1882), but including index for Volumes Nos. 1 to 35, and Nos. 36 to 40, ...	75
VI.	Nine volumes, bound in half-morocco, from No. 1 (1873) to No. 9 (1881), ...	25

Applications should be addressed to Joseph Struthers, Secretary, 29 West 39th Street, New York, N. Y.

The Emmons Research Fellowship of Economic Geology.

The Committee named below has been formed by friends of Samuel Franklin Emmons, late of the United States Geological Survey, to consider the best method of perpetuating his name. It has been decided that the memorial to him shall take the shape of a Research Fellowship, to be known as the Samuel Franklin Emmons Research Fellowship of Economic Geology, which is to be administered by Prof. James F. Kemp, of Columbia University, New York. Subscriptions are invited by his friends to this fund, which the Committee has fixed at $25,000.

Members of the Institute who desire to contribute to this fund will please communicate with the Treasurer, Benjamin B. Lawrence, 60 Wall Street, New York.

The Committee consists of the following:

GEORGE OTIS SMITH, Director, U. S. Geological Survey, Washington, D. C.
H. L. SMYTH, Harvard University, Cambridge, Mass.
JAMES DOUGLAS, 99 John Street, New York, N. Y.
J. A. HOLMES, Director, Bureau of Mines, Washington, D. C.
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H. FOSTER BAIN, 420 Market Street, San Francisco, Cal.
T. A. RICKARD, London, England.
B. B. LAWRENCE, 60 Wall Street, New York, N. Y.

How to Use the "Transactions" of the Institute.

Buy a copy of the Complete Analytical and Alphabetical Index of Volumes I. to XXXV., inclusive; also the new Index of Volumes XXXVI. to XL.

Whether you do or do not own a full set of the *Transactions*, these Indexes will make all of the material contained in the forty volumes available at once without detailed research into each volume separately. Moreover, an easy search will show what particular papers you need to know more about, and perhaps to study. Thus, any person possessing these Indexes can ascertain at once what has been published in the *Transactions* on a given question, and can learn, by writing to the Secretary, what is its nature, whether it is still to be had in pamphlet form, where it can be consulted in a public library, at what cost it can be copied by hand, etc., etc.

In short, to those who own complete sets of the *Transactions*, these Indexes will be a great convenience; but to those who do not, they will be a professional necessity.

The Index Volumes I. to XXXV. is an octavo of 706 pages, containing more than 60,000 entries, duly classified with sub-headings, and including abundant cross-references. The limited edition is becoming exhausted. The new Index, Volumes XXXVI. to XL., supplementing the Index Volumes I. to XXXV., brings the classified references up to the date of Volume XL., June, 1910. Prices: Index Volumes I. to XXXV., bound in cloth, $5; bound in half-morocco, to match the *Transactions*, $6. Index Volumes XXXVI. to XL., bound in cloth, $1.50; bound in half-morocco, $2.50. The delivery charges will be paid by the Institute on receipt of the above price.

LIBRARY.

AMERICAN INSTITUTE OF ELECTRICAL ENGINEERS.
AMERICAN SOCIETY OF MECHANICAL ENGINEERS.
AMERICAN INSTITUTE OF MINING ENGINEERS.
UNITED ENGINEERING SOCIETY.

WILLIAM P. CUTTER, Librarian.

The libraries of the above-named Societies are open from 9 A.M. to 9 P.M. on all week-days, except holidays, from September 1 to June 30, and from 9 A.M. to 6 P.M. during July and August.

The Library contains about 50,000 volumes, including sets of technical periodicals and the publications of scientific and technical societies.

The members of the Institute, with few exceptions, are by the very nature of their profession forced to spend a large portion of their time in localities isolated from sources of information. To such members the Library can render valuable service through correspondence, and letters requesting information will receive special attention. The Library is prepared to furnish references and copies of articles on mining and metallurgical subjects; to determine, if possible, the existence of mining-maps, and to furnish general information as to the geology and mineral resources of all countries as far as these resources are known and published.

It is hoped that the members of the Institute will avail themselves freely of this special service. The Library will welcome inquiries on engineering subjects, and furnish information as far as such information is to be obtained.

All communications should be made as definite as possible so that the information received may be what is desired and not include collateral matter which may not be of interest. In this way the time spent in searching for such collateral matter will be saved, and as a result the information will be sent more promptly and in more usable shape.

The members of the Institute can be of service to the Library by forwarding copies of mining-reports, maps privately issued, and similar material, which will be classified, indexed, and made available to other members.

Suggestions for additions to the Library, either by purchase or personal solicitation as gifts, will be welcomed. It is hoped that members while in the city will use the Library freely, and assurance is given that most careful service will be rendered to them.

Library Accessions.

Sept. 1 to Sept. 30, 1912.

[Copies of the list of additions to the Libraries of the American Society of Mechanical Engineers and the American Institute of Electrical Engineers can be obtained on application to the Secretary of the American Institute of Mining Engineers.]

American Association for the Advancement of Science. Proceedings. Vol. LXII. Washington, 1912. (Exchange.)

American Society for Testing Materials. Proceedings. Vol. XII. Philadelphia, 1912. (Exchange.)

Argentine Republic. Ministerio de Agricultura. Estadistica Minera de la Republica, 1909. Buenos Aires, 1911. (Exchange.)

Atlas Geográfico del Ecuador. By Felicisimo Lopez. N. p., 1907. (Purchase.)

Bibliographie der Deutschen Zeitschriften Literature. By F. Dietrich. Band XXIX. Leipzig, 1912. (Purchase.)

Black Diamond Year-Book and Directory, 1912. Chicago, 1912. (Purchase.)

British Foundrymen's Association. Proceedings, 1910–1911. London, n. d. (Exchange.)

Canada. Commission of Conservation. Annual Report, 3d. Ottawa, 1912. (Exchange.)

Coal Fields of King County. (Bulletin No. 3, Washington Geological Survey.) Olympia, 1912. (Exchange.)

Congreso Cientifico (1° Pan Americano) Ingenieria. Tomo I. Santiago de Chile, 1912. (Gift of Congreso Cientifico (1° Pan Americano).)

Design of Walls, Bins, and Grain Elevators. Ed. 2, revised and enlarged. By M. S. Ketchum. New York, 1911. (Purchase.)

Electrical Code of the City of New York. Comprising the Rules and Regulations for the Construction, Care, Maintenance, and Operation of Ducts, Poles, Conductors, and other Electrical Appliances. 1912. New York, 1912. (Gift of Dr. Joseph Struthers.)

Engineering Standards Committee. British Standard Specification for Cast-Iron Spigot and Socket Soil Pipes. (No. 58.) London, 1912.

——— British Standard Specification for Cast-Iron Spigot and Socket Waste and Ventilating Pipes for other than Soil Purposes. (No. 59.) London, 1912.

——— British Standard Specification for Structural Steel for Bridges, etc., and General Building Construction. (No. 15, revised Aug., 1912.) London, 1912. (Gift of Engineering Standards Committee.)

Escuela de Ingenieros. Memoria anual presentada por la Direccion. Lima, 1912. (Exchange.)

Georgia Geological Survey. Bulletin Nos. 27, 28. Atlanta, 1912. (Exchange.)

Hydrometallurgy of Copper. By W. E. Greenawalt. New York, McGraw-Hill Book Co., 1912. Price, $5 net. (Gift of Publishers.)

[Note.—The extraction of copper by humid methods constitutes a doubly-mixed chapter in metallurgy; that is, it comprises much old as well as much new material; and it involves in practice, almost invariably, roasting, or even smelting, as well as solution and precipitation. Mr. Greenawalt has divided his book into two parts, on Roasting and Hydrometallurgical Processes, respectively, and presents under each head a summary of theory and practice, including the results of his own experimental work as well as the reported experience of others. Since he gives reference to the authorities quoted, and also a good index, as well as a very full table of contents, the book will be highly useful to the students of a subject concerning which trustworthy information is largely fragmentary, and scattered through technical journals.—R. W. R.]

ILLINOIS. BUREAU OF LABOR STATISTICS. Bulletin—Workingmen's Compensation Act, 1912. Springfield, 1912. (Gift of Bureau of Labor Statistics.)

——— Report of Bureau of Labor, 5th. Springfield, 1912. (Gift of Bureau of Labor Statistics.)

ILLINOIS. STATE GEOLOGICAL SURVEY. Bulletin No. 17. Urbana, 1912. (Exchange.)

INDEX OF MINING ENGINEERING LITERATURE. Vol. 2. By W. R. Crane. New York, 1912. (Purchase.)

INTERNATIONAL RAILWAY FUEL ASSOCIATION. Proceedings of the Fourth Annual Convention, 1912. Chicago, 1912. (Gift of International Railway Fuel Association.)

IRON AND STEEL INSTITUTE. Journal. Vol. LXXXV., 1912. No. 1. London, 1912. (Exchange.)

KALENDER FÖR SVERIGES BERGSHANDTERING, 1912. By J. Hyberg. Göteborg, 1911. (Purchase.)

KONIGLISCHEN BERGAKADEMIE ZU CLAUSTHAL. Chronik, 1911-12. Clausthal, 1912. (Exchange.)

——— Verzeichnis der Vorlesungen, 1912-13. Clausthal, 1912. (Exchange.)

KONIGL. SACHS. BERGAKADEMIE ZU FREIBERG. Program, 1912-12. Freiberg, 1912. (Exchange.)

LIMESTONES AND CEMENT MATERIALS OF NORTH GEORGIA. A report on. (Bulletin No. 27, Georgia Geological Survey.) Atlanta, 1912. (Exchange.)

MANCHESTER ASSOCIATION OF ENGINEERS. Transactions, 1911-12. Manchester, 1912. (Exchange.)

MEMORIE DESCRITTIVE DELLA CARTA GEOLOGICA D'ITALIA. Vol. XV. Roma, 1912. (Exchange.)

METALLURGY OF IRON AND STEEL. By Albert Sauveur. Cambridge, 1912. Price, $6. (Gift of Sauveur & Boylston.)

[NOTE.—Prof. Sauveur's splendid book doubtless represents his course on this subject at Harvard University. It is divided into twenty-four "Lessons," separately-paged, and accompanied with two Appendixes—one on Manipulations and Apparatus, the other on the Nomenclature of the Microscopic Constituents of Iron and Steel. Readers of the literature of the new and rapidly progressing science of metallography may well be grateful for such a key to its methods and results. Professor Sauveur's summary fairly fills the place for which it is designed, as "a well-balanced specific and comprehensive treatise," useful alike to students, instructors, manufacturers, and users. It deals with Pure Metals, Pure Iron, Wrought Iron, Low-Carbon Steel, Medium-High and High-Carbon Steel, Impurities in Steel, The Thermal Critical Points of Iron and Steel, Cast Steel, the Mechanical Treatment of Steel, The Annealing, Hardening, Tempering, Cementation, and Case-Hardening of Steel, Special Steels, Cast Iron and its Impurities, Malleable Cast Iron, Constitution of Metallic Alloys, Equilibrium Diagram of Iron-Carbon Alloys, and the "Phase" rule. Under each of these (and sundry subordinate) headings, the different theories are sketched and critically estimated. The impression produced by a rapid examination of the book is that the new science is by no means as confused and incomprehensible as the perusal of many scattered monographs would probably make it appear. Professor Sauveur has not spoken the last word concerning it; but the word he has spoken is clear, wise and helpful.—R. W. R.]

METALLURGY OF TIN. By Henry Louis. New York, 1911. (Purchase.)

MINERAL INDUSTRY. Vol. XX, 1911. New York, 1912. (Purchase.)

MYSORE. GEOLOGICAL DEPARTMENT. Report of the Chief Inspector of Mines, 1910-1911. Bangalore, 1912.

NOTES ON A RECENT STUDY OF THE ATLANTIC CITY STEEL PAINT TESTS. (Bulletin 32, Paint Manufacturers' Association of the United States.) By H. A. Gardner. Philadelphia, 1912. (Gift of Institute of Industrial Research.)

ONTARIO. BUREAU OF MINES. Report, Vol. XXI, pt. 11. Toronto, 1912. (Gift of Ontario Bureau of Mines.)

PHYSIKALISCHE CHEMIE DER METALLE. By Rudolf Schenck. Halle, a/S., 1909. (Purchase.)

PORTLAND CEMENT RESOURCES OF ILLINOIS. (Bulletin No. 17, Illinois State Geological Survey. Urbana, 1912. (Exchange.)

PORTUGAL. COMMISSÃO DO SERVIÇO GEOLOGICO. Communições. Tom VIII. Lisbon, 1910–11. (Exchange.)

PRELIMINARY REPORT ON THE TERTIARY PALÆONTOLOGY OF WESTERN WASHINGTON. (Bulletin No. 15, Washington Geological Survey.) Olympia, 1912. (Exchange)

PROVINCE DE HAINAUT. Rapport sur la Situation de L'Industrie Minérale et Métallurgique, 1911. Frameries, 1912.

PUBLIC ROADS OF GEORGIA, THIRD REPORT. (Bulletin No. 28, Georgia Geological Survey.) Atlanta, 1912. (Exchange.)

LE PYROMETRE THERMO-ELECTRIQUE. By H. Pécheux. Paris, 1909. (Purchase.)

RUSSIA. MINES DEPARTMENT. Iron and Coal Industry of Russia in 1896. (In Russian.) St. Petersburg, 1898. (Purchase.)

SILVER-LEAD LODES OF THE WARATAH DISTRICT. (Report No. 2, Tasmania Geological Survey. Hobart, 1911. (Exchange.)

SOCIÉTÉ GÉOLOGIQUE DU NORD. Annales. Tome XL, 1911. Lille, 1911. (Exchange.)

SOCIÉTÉ DE L'INDUSTRIE MINÉRALE. Annuaire, 1912–13. Saint Etienne, 1912. (Exchange.)

SVENSKA TEKNOLOGFÖRENINGEN. Ledamotsförteckning jämte Stadgar och Ordningsregler, Juli, 1912. Stockholm, 1912. (Exchange.)

SYLLABUS OF MATHEMATICS. A Symposium Compiled by the Committee on the Teaching of Mathematics to Students of Engineering. Ithaca, 1912. Price, $0.75. (Gift of Society for the Promotion of Engineering Education.)

[NOTE.—This little book is the work of a Committee appointed at a joint meeting of mathematicians and engineers, held at Chicago in December, 1907. The Committee was directed to report to the Society for the Improvement of Engineering Education, and this report was accepted by that Society at its 19th annual meeting, Pittsburg, June, 1911. The members of the Committee are instructors or engineers of high standing, and they have presented here the preliminary and tentative draft of part of "a synopsis of those fundamental principles and methods of mathematics which should constitute the minimum mathematical equipment of the student of engineering." The book contains a syllabus of each of the following subjects: elementary algebra, elementary geometry and mensuration, plane trigonometry, analytic geometry, differential and integral calculus, and complex quantities. The subject of dynamics is apparently reserved for future similar treatment. The work is neither a text-book nor a hand-book, but a summary of the leading principles which the student ought to know and remember—that is, a guide by which his course should be planned, and a measure by which its accomplishment may be tested. It is a valuable contribution to the simplification and unification of mathematical teaching.—R. W. R.]

TASMANITE SHALE FIELDS OF THE MERSEY DISTRICT. (Bulletin No. 11, Tasmania Geological Survey.) Hobart, 1912. (Exchange.)

TENNESSEE. STATE GEOLOGICAL SURVEY. Bulletin No. 14. Nashville, 1912. (Exchange.)

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- Bulletin No. 118. Chicago Giant Rock-Drills—Auxiliary Valve Type. 8 pages.
- Bulletin No. 120. Instructions for Setting Up and Operating Chicago Giant Rock-Drills. 8 pages.

GENERAL ELECTRIC CO., Schenectady, N. Y.
- Bulletin No. 3191. Drum Type Reversing Switches. 12 pages.
- Bulletin A 4,001. Oil Switches for Small and Isolated Plant 3,300-volt Alternating-Current Service. Type F, form K 13.
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- Bulletin No. 4,968. Electric Automobile Appliances.
- Bulletin No. 4,995. Direct-Current Switchboards.
- Bulletin No. 4,996. Alternating Current Switchboard Panels.
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- Catalogue No. 75. Water Lifted by Compressed Air. 1912. 67 pages.
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MESTA MACHINE Co., Pittsburg, Pa. Brief Description and Illustrations of the Plant and Production. 47 pages.

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Titanium Rail Tests on Delaware & Hudson Co.
Rail Tests on Lehigh Valley R. R.

MEMBERSHIP.

NEW MEMBERS.

The following list comprises the names of those persons elected as members who accepted election during the month of September, 1912:

Members.

KUZELL, CHARLES R., Met., Anaconda Copper Mining Co.,
Boston and Montana Reduction Dept., Great Falls, Mont.
LONGAN, WALKER B., Min. Engr................P. O. Box 39, Breckenridge, Colo.
LOWER, JOSEPH B., Supt., Chintui and Candlestick Camps, of
The Oriental Consolidated Mining Co., Unsan, Chosen, Korea.
PRÖMMEL, HARALD W. C., Mine foreman, 1a Calle Madrid No. 15,
Mexico City, Mexico.
SCHUMACHER, WILHELM, Engineer, Haberland Str. 2, Berlin, W. 30, Germany.

CANDIDATES FOR MEMBERSHIP.

The following persons have been proposed during the month of September, 1912, for election as members of the Institute. Their names are published for the information of members and associates, from whom the Committee on Membership earnestly invites confidential communications, favorable or unfavorable, concerning these candidates. A sufficient period (varying in the discretion of the Committee, according to the residence of the candidate) will be allowed for the reception of such communications, before any action upon these names by the Committee. After the lapse of this period, the Committee will recommend action by the Council, which has the power of final election.

Members.

Charles A. Baechtold, New York, N. Y.
Proposed by Louis D. Huntoon, Edward B. Sturgis, George F. Kunz.
William Walley Davis, Steelton, Pa.
Proposed by Edward N. Trump, E. C. Witherby, W. H. Blauvelt.
Harry Patton Dowler, Heilwood, Pa.
Proposed by John McLeavy, Thomas A. Furniss, Thomas Fisher.
Herbert C. Ebos, Mexico City, Mexico.
Proposed by R. M. Raymond, A. F. Main, Victor M. Braschi.
Fontaine Maury S. Gillespie, Abangarez, Costa Rica, C. A.
Proposed by Percy L. Fearn, H. T. Purdy, J. C. Porter.
Frank Watrous Hamilton, Brooklyn, N. Y.
Proposed by Louis D. Huntoon, John H. Lidgerwood, George F. Kunz.
Karl Frederick Hoffman, Mexico City, Mexico.
Proposed by A. Monell, W. A. Bostwick, W. W. Mein.
Zay Jeffries, Cleveland, Ohio.
Proposed by Charles H. Fulton, A. W. Smith, Frank R. Van Horn.
George Basil McCunn, Wabana, Newfoundland, Canada.
Proposed by R. E. Chambers, Bradley Stoughton, Louis D. Huntoon.
L. L. Middlekamp, Ellamar, Alaska.
Proposed by G. M. Gouyard, H. A. Keller, Lewis A. Levensaler.

Robert P. Millard, Cleveland, Ohio.
Proposed by H. Alfred Millard, H. C. Hale, G. F. Knapp.

Edwin Morris Mohrman, Twin Falls, Idaho.
Proposed by G. R. D. Bailey, Robert N. Bell, Henry Kehoe.

J. Burns Read, Cleveland, Ohio.
Proposed by Charles H. Fulton, Frank R. Van Horn, A. W. Smith.

Charles Prentice Turner, Steelton, Pa.
Proposed by Joseph W. Richards, Henry S. Drinker, Howard Eckfeldt.

Sidney Rolle, Chrome, N. J.
Proposed by Lawrence Addicks, Robert N. Draper, John N. Judson.

James Arthur Singmaster, Palmerton, Pa.
Proposed by J. A. Van Mater, A. Heckscher, George C. Stone.

Charles Sumner Smith, Globe, Ariz.
Proposed by James Douglas, George B. Lee, A. R. Ledoux.

Albert Edward Wiggin, Anaconda, Mont.
Proposed by William Wraith, C. D. Demond, C. W. Goodale.

Associate.

André Marie Joseph Camille Barthelemy, Carthagena, Spain.
Proposed by Godfrey T. Vivian, Walter McDermott, A. Ernest Thomas.

CHANGES OF ADDRESS OF MEMBERS.

The following changes of address of members have been received at the Secretary's office during the month of September, 1912. This list, together with the lists published in *Bulletin* Nos. 63 to 69, March to September, 1912, and the foregoing list of new members, therefore, supplements the annual list of members corrected to Feb. 1, 1912, and brings it up to the date of Oct. 1, 1912.

AGNEW, JOHN A., Care Union Bank of Australia, Ltd., London, E. C., England.
ALDRIDGE, WALTER H.......14 Wall St., New York, N. Y.
AMSDEN, O. B., Min. Engr.......325 E. Iliff Ave., Denver, Colo.
BACKUS, GEORGE S.......Elk City, Idaho.
BEHR, HANS C., Gold Fields American Development Co., Ltd., 71 Broadway, New York, N. Y.
BLACK, ROBERT M., Asst. Prof. of Coal-Mining, University of Pittsburgh, Pittsburgh, Pa.
BOEHMER, MAX.......419 Jacobson Bldg., Denver, Colo.
BOLLES, J. H.......114 W. 79th St., New York, N. Y.
BOSCH, ANTON.......Instructed to hold all mail.
BURT, CHARLES S.......936 S. Summit St., Iowa City, Iowa.
CABOT, JOHN, JR.......Care A. M. Hoyt & Co., 1 Broadway, New York, N. Y.
CHANEY, R. GORDON, JR., Engr., Independent Coal & Coke Co., Kenilworth, Utah.
COX, W. RAY.......512 Custom House Bldg, San Francisco, Cal.
CREWE, LEONARD C., Genl. Mgr., La Follette Iron Co., La Follette, Campbell Co., Tenn.
CRUM, J. RICHMOND.......Trentham, Stoke-on-Trent, England.
CURRIE, JOHN M.......Care Lake & Currie, Norfolk House, Laurence Pountney Hill, London, E. C., England.
DAKE, WALTER M, JR.......The Bully Boy Co., Maryvale, Utah.
DANE, EDWARD G.......Perak, Federated Malay States.
ELDRED, BYRON E., Prest.......Commercial-Research Co., Tuckahoe, N. Y.
EMMONS, CAROLUS D.......156 E. 11th St., Eugene, Ore.
EYE, CLYDE M.......Concheño Mining Co., Temosachic, Chih., Mexico.
FAY, ALBERT H.......Bureau of Mines, Washington, D. C.
FISHBURN, RANDOLPH E.......Schrader Bldg., Tucson, Ariz.
FULTON, CHESTER A.......University Club, Mexico City, Mexico.
FURNESS, DWIGHT, Prest, Dwight Furness Co., Apartado 2, Guanajuato, Mexico.
GAHL, RUDOLF.......Care Copper Queen Cons. Copper Co., Douglas, Ariz.

Hall, George..........12 Sheepcote Road, Harrow, England.
Hopper, Walter E..........Michigan College of Mines, Houghton, Mich.
Hunt, Archibald J..........15 Rue de Beucker, Antwerp, Belgium.
Hutcheson, William C..........Wind Rock, Tenn.
Jennings, Hennen..........2221 Massachusetts Ave., Washington, D. C.
Keiller, Alexander G..........Aguacate Mines, San Mateo, Costa Rica, C. A.
King, Paul S...Care Chas. McFadden, Jr., 402 Arcade Bldg., Philadelphia, Pa.
Kirchhoff, Charles..........587 Riverside Drive, New York, N. Y.
Kruttschnitt, Julius, Amer. Smelting & Refg. Co., Fenner Bldg., Tucson, Ariz.
Lane, Henry M., Cons. Fdy. Engr. and Met., Prest., H. M. Lane Co.,
18 Piquette Ave. E., Detroit, Mich.
Leland, Frank M., Prest., and Act. Mgr., Empire Copper Co., Mackay, Idaho.
Linville, Clarence P., Supt. Blast Furnaces for Joseph E. Thropp, Everett, Pa.
McAllister, J. Edgar..........15 King St., W., Toronto, Ont., Canada.
McDevitt, James E..........14 Taconic St., Pittsfield, Mass.
McEwen, Atholl F., Care C. G. McEwen, West Rand Cons. Mines,
Krugersdorp, Transvaal, So Africa.
McMaster, Alexander T. C..........659 Spadina Ave., Toronto, Ont., Canada.
Marriott, Frederick A., "Roslyn," Summer St., Orange, N. S. W., Australia.
Martin, Samuel G..........Martin Min. & Power Co., Eureka, Colo.
Mitchell, Deane P..........1 London Wall Bldgs., London, E. C., England.
Mitke, Charles A..........Box 1369, Bisbee, Ariz.
Nichol, William..........Care African Banking Corpn., Kimberley, South Africa.
Nissen, Peter N., Care Head, Wrightson & Co.,
5 Victoria St., Westminster, London, England.
Nobs, Frederick W..........La Leonesa, Matagalpa, Nicaragua, C. A.
North, Wheeler O..........303 Henne Bldg., Los Angeles, Cal.
Oster, William T..........Bonanza, Idaho.
Pearson, William R..........Instructed to hold all mail.
Richards, H. DeC..........Crocker Bldg., San Francisco, Cal.
Rule, J. Arthur..........Rio Tinto Copper Co., Terrazas, Chih., Mexico.
Scholz, Carl..........911 La Salle Sta., 144 Van Buren St., Chicago, Ill.
Schwarz, Charles E..........6029 Maple Ave., St. Louis, Mo.
Severy, Clarence L..........385 Oakland Ave., Pasadena, Cal.
Short, Frank R..........1043 Monadnock Bldg., San Francisco, Cal.
Somerset, Harry St. J., Jr., Care J. B. Wilson, Hunter's Hill,
Sydney, N. S. W., Australia.
Staunton, William F.......... 512 S. Harvard Blvd., Los Angeles, Cal.
Straw, Charles A..........218 Warren St., Hudson, N. Y.
Taber, Stephen..........University of South Carolina, Columbia, S. C.
Thomae, W. F. A., Care Mazapil Copper Co, Apartado 64, Saltillo, Coah.,
Mexico.
Thomas, David R., Mgr..........Predilecta Mining Co., Guanacevi, Dur., Mexico.
Torrey, Herbert G..........10 Wall St., New York, N. Y.
Turner, Albert A..........Beatty, Nev.
Van Arsdale, George D..........Copper Queen Cons. Mining Co., Douglas, Ariz.
Van Siclen, Matthew..........Turtle Creek Siding, Albert Co., N. B., Canada.
Vigeon, Edward C., Asst. Genl. Mgr., Spassky Copper Mine, Ltd.,
Karagandy, Akmolinsk District, Siberia.
Webster, Paul W., Care Lutz-Webster Engineering Co.,
3103 Grays Ferry Road, Philadelphia, Pa.
Wilkinson, William F..........Hurstbourne Priors, Whitchurch, Hants, England.
Wolf, Albert G..........Gardnerville, Nevada.

ADDRESSES OF MEMBERS AND ASSOCIATES WANTED.

Name.	Last Address of Record, from which Mail has been Returned.
Bates, Mowry,	Hughes Porcupine Mines, Ltd., Porcupine, Ont., Canada.
Baxter, Francis K., Jr.,	423 Wells Fargo Bldg., San Francisco, Cal.
Cook, Edward H.,	Minas Birimoa, S. A., Birimoa, via Canelas, Dur., Mexico.
Danforth, A. H.,	Cotopaxi, Colo.
David, W. M.,	Care C. G. Hussey & Co., Second Ave., Pittsburg, Pa.

Dyott, George M., ... 50 Church St., New York, N. Y.
Fergusson, Hugh B., ... 216 Loo Bldg., Vancouver, B. C., Canada.
Fitzgerald, Thomas F. M., ... 211 Sharon Bldg., Salt Lake City, Utah.
Furness, James W., ... Coffee, Trinity Co., Cal.
Gmelin, Ernest, ... Apartado 57, Aguascalientes, Mexico.
Higgins, Edwin, ... Randsburg, Cal.
Hilson, Cleaveland, ... Livingston, Mont.
Hollis, R. W., ... Silverton, Colo.
Johnson, Dion L., ... 325 Water St., Pittsburg, Pa.
Lampshire, John O., ... Vulture Mine, Wickenburg, Ariz.
Leavell, John H., ... Buffalo Mine, Cobalt, Ont., Canada.
Le Noir, Frank H., ... Box 16, Mt. Bullion, Cal.
McDougall, Wallace D., ... 20 Bedford Place, Russell Sq., London, Eng.
Mitinskij, Alexander N., ... V. O. 12, line 33, St. Petersburg, Russia.
Moore, Roy W., ... P. O. Box 48, Velasco, Tex.
Munroe, Martin, ... Bengal Coal Co., Murulidih, Mohada, B. N. Ry., Bengal, India.
Nelson, D. W. C., ... Baker City, Ore.
Oldfield, Frank W., ... Mexican Mines Co., Bolaños, Jal., Mexico.
Perks, Harry B., ... 419 Board of Trade Bldg., Portland, Ore.
Peterson, Frank, ... H. W. Hellman Bldg., Los Angeles, Cal.
Rathborne, Merwyn R. W., . . . Amargosa, via Las Vegas, Nev.
Rhew, James W., Cia. Minera y Exploradora de Ventanas, S. A., Ventanas, Dur., Mex.
Rickard, Harold, ... Foley-O'Brien, Ltd., So. Porcupine, Ont., Canada.
Sheldon, Waldo, ... Urique, Chih., Mexico.
Thornton, Edward T., ... Apartado 30, Matehuala, S. L. P., Mexico.
Twynam, Henry, . . . O. K. Copper Mine, Cairns, No. Queensland, Australia.
Vinicombe, Robert E. B., ... Post Restante, Vladikafkas, So. Russia.
Watson, Ralph W., ... Calloo, Utah, Clifton Mail box.
Webster, Erastus H., ... Hotel Cosmopolita, Guadalajara, Jal., Mexico.
White, W. P. . . . . 10 Bernard St., Spokane, Wash.

NECROLOGY.

The deaths of the following members were reported to the Secretary's office during the month of September, 1912:

Date of Election.	Name.	Date of Decease.
1895.	**Chalmers, John A.,	September 9, 1912.
1872.	*Kent, Joseph C.,	August 31, 1912.

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* Member. ** Life Member.

[TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS.]

The Occurrence of Gold in the Eocene Deposits of Texas.

BY E. T. DUMBLE, HOUSTON, TEX.

(Cleveland Meeting, October, 1912.)

For many years there have been occasional reports of the discovery of gold from a belt of the coast country of Texas which is underlain by deposits belonging to the lower Eocene. For the most part they have been treated as newspaper reports and no examination or record of them has been made. Among such, it may be mentioned that in 1896 there was a flurry over reported gold-discoveries west of Luling and elsewhere in this belt, and in 1903 some prospect-shafts were sunk on the Gonzales branch of the Galveston, Houston & San Antonio railway, 3 miles below Harwood. During 1912, further discoveries have again brought the matter to our attention, and it may be of interest to record the facts as we have found them.

Some years ago specimens were sent me, which were said to have been taken from certain beds on Gazley creek, and concentrated by washing in a rocker. In them were several particles of gold sufficiently large to be visible to the eye. Under the microscope they showed to be, with few exceptions, sharp cornered, and not rounded as is usual with placer-gold. They were not filings, however, but their sharpness seemed rather to be due to a recent freeing from their matrix. One or two grains were rusty, and one copper-hued, but the most of them were tolerably bright.

On examining the material of the samples small grains or fragments were found, of a yellowish brown color, which seemed to be of a finely-granular structure—possibly a sandstone. In these fragments I found particles of gold, one of them being large enough to be visible without a lens.

Gazley creek empties into the Colorado river just north of Smithville, and for a few miles north of that place its banks show, beneath the deposits of more recent materials, the sands and clays of the Lignitic division of the Eocene, with lignite, shells of iron, etc.

A characteristic section is as follows:

Bed No.		eet.
1.	Detritus covered,	15
2.	False-bedded sands with iron, the seams and lamina being indurated ferruginous sandstone,	4
3.	Similar to No. 2, but less hardened and containing some seams of chocolate clay,	3 to 4
4.	Lignitic sands and clays with some lignite,	1 to 3
5.	Gray clay with shells of iron, to water,	1 to 3

It was from these beds that the specimens sent me were supposed to have come.

I spent two days along the creek securing representative and average samples from each of the beds and in panning dirt from various parts of the beds and from the creek. In panning, only a very few colors were seen, and these were from beds Nos. 3, 4, and 5. I also found in the beds fragments of sandstone similar to those in which the gold occurred in the original samples, although I did not find gold in any of my sandstone fragments.

Very careful assays were made of the samples I collected, with the following results:

Sample No.	Location.	Gold.
1.	Upper part bed No. 3,	None.
2.	Middle part bed No. 3,	Trace.
3.	Lower part bed No. 3,	Good trace.
4.	Bed No. 4,	Good trace.
5.	Bed No. 5,	Faint trace.
6.	Iron-seam, bed No. 5,	None.
7–*a*.	Iron-seam, bed of creek,	Strong trace.
7–*b*.	Iron-seam, bed of creek,	None.

The only visible gold in the samples taken by myself was found in No. 4, in which several small particles were clearly seen under the lens.

The actual occurrence of gold in these deposits was therefore proved, although I was unable to find it in such quantity as was indicated by the original samples.

The work south of Harwood was brought about by finding in wells in that vicinity a stratum of fine argillaceous sand mixed with glauconitic matter and carrying quantities of small crystals of pyrite which showed gold when assayed.

The sand-bed, reached at 200 ft., was 80 ft. in thickness.

Assays of carefully-taken samples made by reputable assayers showed values of $20 and upward. Two shafts were started and were sunk to more than 100 ft. in depth, but, as I understand it, owing to trouble with quicksand below that depth they were abandoned before the supposed gold-bearing sand was reached.

Later, parties digging a well southwest of Harwood were attracted by the appearance of the materials found, and sent samples to the U. S. Assayer at El Paso, who reported very favorably on them. The well was then enlarged and deepened, and carefully sampled. The assays, made of materials taken every foot from the bottom upward for 38 ft., all showed the presence of gold, although only as traces in six specimens. In the others the gold-values ranged from 16 cents to $8 per ton, with small amounts of silver additional.

Samples of these sands from other wells in this vicinity also show small values in gold.

In both these places the geological relations are the same as those in the Gazley Creek region, so that all occurrences which have come under our notice are connected with beds of the Lignitic stage (Chickasawan) of the Eocene.

The materials of the Lignitic beds in this area are interbedded clays and sands with beds of lignite. The sands are quartzitic, more or less micaceous and carry occasional small pieces of feldspar. They also carry a considerable quantity of iron in the shape of scattered grains and nodules of pyrites and some ferric oxide.

The composition of these beds immediately suggests the granite area of the Llano region as the place of origin of the materials entering into them, and the channel of the Colorado river or its early Tertiary predecessor as the route by which they could have reached the shores of the Lignitic lagoon or swamp in which they were deposited.

Gold and silver are known to occur in the porphyries and micaceous schists of the Llano region, and gold is found among the sands lying in the creeks flowing over or through these rocks. Considerable quantities of placer-gold have been found in the Big Sandy, and it is found in the sands of the Colorado as far south as Austin.

The later Tertiary deposits of the Colorado basin south of Lagrange show quantities of similar materials derived from the Llano region, the fragments of feldspar and granite being of much larger size than any that enter into the Lignitic beds. Consequently, while grains of free gold might not have been transported such distances as are here involved, fragments of the sandstones, porphyries, or mica-schists with gold inclusions might have been so transported and by subsequent disintegration have freed the gold.

It would, therefore, seem entirely probable that the Llano region, with its granites and porphyries, and its gold-, silver- and iron-ores, was largely uncovered in early Eocene times and was subjected to active erosion, the detritus from it being swept coastward by waters running into the Colorado and deposited as the beds we now call the Lignitic.

These deposits are now known to carry gold locally over a region about 60 miles in length, and while it may well be that the colors do not exist at any place in so concentrated a condition as to make it possible to work them, the fact that the gold does actually occur will doubtless induce repeated attempts to do so. Furthermore, the nature of the occurrence is of sufficient interest to warrant a much more extended study than has yet been given it.

SUBJECT TO REVISION.

[TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS.]

Present Conditions of Mining in the District of Vladivostok, Siberia.

BY ALBERT F. J. BORDEAUX, THONON-LES-BAINS, FRANCE.

(Cleveland Meeting, October, 1912.)

THE immediate vicinity of the sea-shore, affording special facility for the exportation of ores, makes it possible to work certain mines in the Vladivostok district, which, in more remote places of Siberia, would remain unproductive. Outside of the city of Vladivostok the population is small and the country is very little prospected. Additional discoveries of valuable deposits like the large body of calamine which has recently been opened in the neighborhood of earlier worked lead-silver ores, in the Tetiuhe valley, are still in the field of possibilities.

For many years gold alone was produced, in part from the coastal rivers and in part from a quartz-vein on Askold island near Vladivostok. But the production is diminishing, and prospectors are looking for other rivers in the interior, such as the tributaries of the Ussuri. These rivers are now more accessible than formerly, due to the slow but gradual development of colonization by the Russian peasants. The few natives of the country, called Orodjones, are of the Chinese race. They resemble the North American Indian and are hardy fur-hunters. They have given some attention to placers and have already shown good specimens of gold-dust from the rivers Iman, Bikin, Noto, etc.

Coal-mining was begun in the Vladivostok region more than 20 years ago, but in a somewhat irregular way. The industry is not as yet important and the best coal-seams are badly prospected, or are even neglected.

Copper, lead, and iron exist in several places. The amounts give some encouragement to the anticipation that the discoveries will give rise to important metallurgical works.

I here present a review of the various deposits, based upon such safe information as is in hand, when I have not drawn upon my personal observations. Fig. 1 is a map of Eastern Siberia, showing the mineral deposits.

Gold.

During the past two years, the quantity and value of the gold-production of the Ussuri, or Vladivostok, district has been as follows. The area included extends north only as far as the Tumnine river, 600 miles from Vladivostok.

	Quantity.		Value.	
	1909. Kilograms.	1910. Kilograms.	1909. Per Ton.	1910. Per Ton.
Tumnine river, ...	44.611	11.774	$1.20	$2.40
Nikolaevsk-Tumnine, . .	80.438	20.260	1.50	1.95
Askold, sands, ...	0.433	2.524	0.40	0.18
Askold, quartz, ...	15.994	6.185	3.95	1.25
Ussuri bay, ...	0.911		0.55	
Total, ...	142.387	40.743	for 77,652 tons.	

The production is small, owing to the present exhaustion of the coastal rivers of easy access. The mining, moreover, is done in the most primitive way. For the quartz of Askold, there is a 5-stamp battery. The total production of this mine for 30 years is about 850 kg., valued at $500,000.

Lately, prospecting has begun on the Iman river, and as a result, 7.5 kg. of gold has been obtained from 2,044 tons of ore. Gold exists in the small creeks, and also in the Iman itself, and on the terraces of a probable glacial formation, that borders the river. Dredging, as well as hydraulicking, may in time be employed, since the surrounding hills, more than 300 m. high, give sufficient fall for hydraulicking purposes.

The natives have found gold in the Bikine, Noto, and Khor rivers. There is gold also along the Manchurian boundary.

There are some quartz veins on the coast, opposite Askold, and also at Sedimi and Adimi, on the other side of the Amoursky zaliv (gulf), 40 miles from Vladivostok. The veins contain arsenical pyrite, and antimonial copper sulphides, in a gangue of schists. Large outcrops of hornblende-granite, and a black igneous rock, probably a gabbro, occur close by.

Calamine, or Zinc-Ores.

The discovery of calamine in Far-Eastern Siberia was made by me, near an old Chinese lead-silver mine. The old diggings were about 10 m. deep and very irregular. Probably the Chinese had a way of their own of separating lead from silver by oxidizing the lead sulphides. During the smelting-process the greater part of the lead seems to have been lost.

Prospecting had been done by Bryner & Co., of Vladivostok, since 1897. This firm, in 1902, showed me a few specimens of

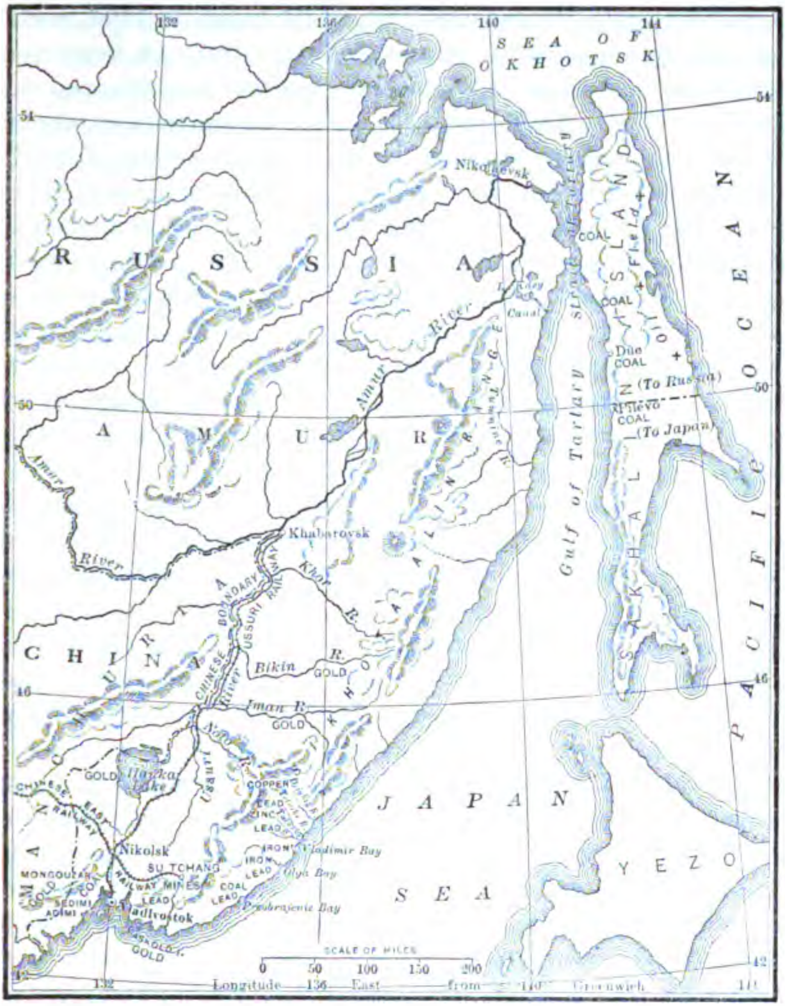

FIG. 1.—MAP OF EASTERN SIBERIA, SHOWING MINERAL DEPOSITS.

ore, from which I inferred the existence of calamine. Not until 1903 was I able to inspect the place, and then the discovery was an easy matter. Only one familiar with mineralogy would have suspected the value of this white rock.

The Tetiuhe deposit is about 250 miles from Vladivostok by sea, and 21 miles from the sea-shore. The distances are such that with sufficient tonnage mining may be profitably carried on. After two or three months of prospecting, more than 10,000 tons of ore with 48 per cent. of zinc was demonstrated.

Unfortunately, the Japanese war, which began in 1904, made the resumption of work impossible until 1907. A road was then built and about 900 tons of calamine was shipped to Europe. During 1908 and 1909, a 2-ft. gauge railway, about 36 km. long, was constructed, by the firm of Arthur Koppel. The loading-facilities in Titiuhe bay were improved by a pier. At the mines, according to Russian custom, schools, hospital, bath-house, library, and many other buildings were erected. A double-track gravity cable-tram joined the mine and the railway. The telephone and telegraph connected all departments and the manager with Olga bay, and thus with Vladivostok. There are more than 1,000 workingmen, of whom two-thirds are Chinese and Coreans. At Titiuhe is the largest mining-camp in the Vladivostok district.

By the end of 1911, more than 55,000 tons of calamine, averaging 43 per cent. of zinc, had been shipped. There is still from 35,000 to 50,000 tons of ore in sight, probably containing more than 40 per cent. of zinc.

Like similar calamine-deposits, the Tetiuhe ore appears in lenses and masses of irregular shape in a metamorphic limestone. Such lenses are called *griffons* in Greece, because they extend like fingers into the inclosing rock. It is a metasomatic, or replacement, deposit. The zinc carbonate has replaced the calcium carbonate by the reaction of acid solutions containing the zinc sulphide, or blende, upon the limestone. The zinc-blende and galena adjoining the calamine constitute a contact-deposit along the border of a huge porphyry sheet with underlying limestone. In the porphyry itself, there are some lenses of the sulphides, galena, zinc-blende, and pyrite. Apparently some silicic acid derived from the porphyry has given rise to some zinc silicate, which is mixed with the calamine. As copper is present, the calamine is sometimes tinted with the blue carbonate and the red oxide of copper, so that it resembles the Mexican *herrerite*.

Among the metasomatic minerals produced along the contact of the porphyry, there is a large quantity of tremolite, crystallized in the shape of needles radiating from a center. It is very hard, and has various colors, from red to green, according to the proportion of iron present. After long exposure, the tremolite undergoes decomposition and becomes very soft, and being intermixed with galena and zinc-blende, it is a serious drawback in the separation of the sulphides. Among the contact rocks, on the top of the deposit, there are also some quartzites and some black metamorphic slates.

The porphyry is a quartz-porphyry with biotite as the dark silicate, and with some crystals of zinc-blende. It has a dark-green color, is very compact, and often looks like andesite or one of the more recent forms of dioritic rocks. On the upper parts of the hill, there is a gray porphyry which seems to be an alteration-product of the green variety.

The limestone is Jurassic in age. Fossils are scarce, yet a *Cidaris* and some mollusks have been found. The latter have not been accurately determined. The Jurassic beds run parallel with the Sikh-ot-alin range, but with many interruptions. It might be of interest to prospect this range of mountains to the north towards Saghalien island, and to the south along the intersection with the sea-shore. In the latter area the calamine-deposits, if there are any, would be more advantageously situated. But so far, outside Tetiuhe, zinc carbonate has not been discovered.

There is a small deposit of zinc silicate near Olga, 50 miles south from Tetiuhe. This is also a contact-deposit in connection with iron-ore instead of lead-ore, and it will be more properly described with the iron-deposits. The quantity does not seem to exceed 2,000 or 3,000 tons of zinc-ore.

From the above description, it may be inferred that there is a possibility of more discoveries of calamine along the Coast range, as was the case with similar deposits in Sardinia, Tunis, Mexico, and elsewhere. If found, their development and mining would be much helped by the experience gained on the Tetiuhe mines. It may be said here that the freight to Europe from the Tetiuhe bay is only 25 shillings, or $5, per ton. It may also be remarked as an important consideration, that the price of calamine in Europe (Antwerp, Hamburg, or Dunkirk) fluctuates with the market: a rough ore with 43 per cent. of zinc, and with the zinc at £25 a ton, sells at £5.10, or about $27.50, per ton.

Although the lead and zinc sulphides of Tetiuhe are part of the same deposit, they will be described separately, as being entirely different ores and requiring other methods of mining and treatment.

Silver-Lead Ores.

The main silver-lead deposit at present is the Tetiuhe mine, Fig. 2. The silver-ore was the occasion of the discovery of the calamine. Prospecting-work dates back to 1897, but the mine was developed only after 1907. The ore is an intricate aggregate of the sulphides—galena, zinc-blende, pyrite and gray copper, in a gangue of tremolite (actinolite). The different constituents are so closely mixed that no hand-sorting is possible. The proportion, moreover, of silver does not vary so much with the lead sulphides as with the copper sulphides (or gray copper), and as this last constituent is scarcely visible, being so scarce, no separation can be made except after fine crushing.

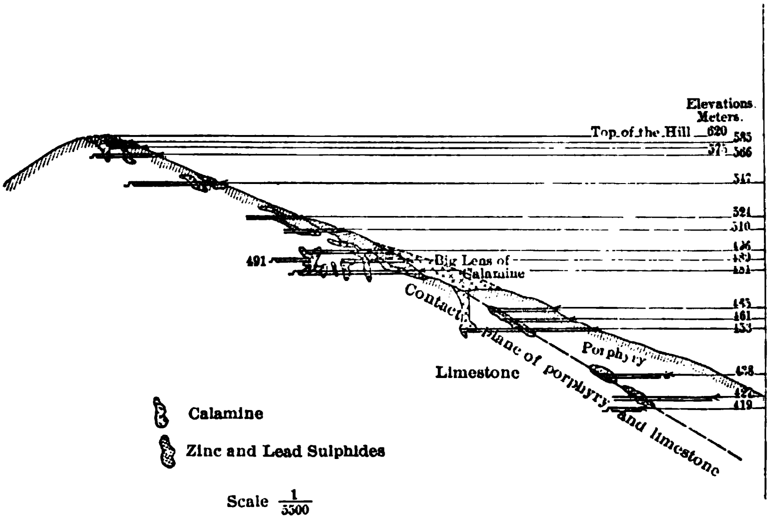

FIG. 2.—SKETCH OF THE MINE AND WORKS OF TETIUHE (SIBERIA) ZINC- AND LEAD-DEPOSIT.

I have previously described this deposit as a contact between the limestone and a blanket of porphyry which appears towards the outside of the hill, making possible the opening of the ore by short adits. The contact-lenses are most developed, but in the lower part of the mine there are, besides, a few segregations of sulphides in the igneous magma itself (magmatic inclusions). The constituents are the same, so that one may infer their close relations to the contact-lenses. A deposit of this kind is of course very irregular, but the type is not scarce in Siberia, where many lenses of lead-silver ores have been worked in the Altai and Nertchinsk districts, as will be described hereafter.

The total amount of underground works at Tetiuhe for developing the calamine, and simultaneously the lead-ores, is about 4 km. of tunnels in 16 levels on a total elevation of 170 m. These figures include neither the shafts and stopes, nor the big open-cuts for the calamine. All the great stopes are practically development-work for the lead-silver ore-bodies.

The total amount of sulphides thus developed may be estimated roughly at between 200,000 and 250,000 tons, and the question at present is how to make the most of them. The first experimental runs for a concentrating-plant have been made in Europe, with the ultimate plan of erecting a plant at the mouth of the Tetiuhe river. The treatment of various similar deposits in the neighboring valleys of Tatushe, Olga, Preobrajenie, Djighit, etc., would eventually give rise to a custom concentrating-plant, as in many districts of the United States and Mexico.

There are other lead-silver deposits of the same kind in Siberia. The one most closely similar, and also the nearest, is that of Nertchinsk, where some calamine, or zinc silicate, was discovered long ago. The total number of mines at Nertchinsk has reached 448, of which 306 are in the limestone. There were 14 main deposits. The largest have produced 170,000 tons. The average proportion of lead has been 2.69 per cent., and the silver 283 g. (9⅛ oz.) per ton, in a total production of 1,457,000 tons, extending over 167 years, from 1703 to 1870. Recently the production has fallen to an almost insignificant amount.

It is not at all impossible that the district of Vladivostok may also become an important producer of lead, with the eventual erection of a smelting-plant at some convenient place.

It should be noted that the smelting-furnaces of small capacity, which could be erected in former times, are now no longer possible. The cost of labor is much higher, while at the same time the value of lead and silver has greatly diminished. Plants of large capacity, although expensive to build, can be worked economically when the general expenses are distributed over a correspondingly large tonnage of ore.

Copper-Deposits.

Copper was known to exist in the Vladivostok district as early as 1902, at the time of my first visit, but even now, in 1912, the amount of prospecting is small. Yet the copper-deposits are really of interest. In some places they are true fissure-veins in andesite. In the Djighit valley, north of Tetiuhe, they are not far from the sea. The thickness of the veins is about 2 ft., and the vein-matter is quartz and copper pyrite. The location, about 100 m. above the river, is very favorable, and the proportion of copper seems high.

There are also some copper-ores (gray copper) in the limestone of the Tetiuhe valley, along the contact with the porphyry. Some copper-ore also appears in the lead-zinc ores of Tetiuhe. Such deposits, while less interesting than veins, may yet possess importance should the copper-smelting industry take root in the district.

There is a premium of £12, or $60, per ton of copper produced in Russian territory, and there is also a premium of £4. 5*s*., or $21, per ton of lead. This award was evidently made to encourage the smelting industry in Eastern Siberia.

There is some iron-ore near Nicolaevsk, at the mouth of the Amur river, but the locality is outside of our district. In the Vladivostok district, there are two iron-deposits, a short distance from each other, the one near Olga bay, and the other near Vladimir bay. They are segregation- and contact-deposits, in and between granite or porphyry and limestone. The latter has been metamorphosed to marble, with a thin crust of calamine, or zinc silicate, which must have been derived from small patches of zinc sulphide in the vicinity.

The ore is magnetite, either pure or mixed with garnet and actinolite or tremolite, the same contact-minerals which appear at Tetiuhe. Titanium also was found in a few analyses. That there are some sulphides in the neighborhood, I infer from the Chinese shafts, which have not yet been reopened, but which were very likely sunk for silver-lead ores. This magmatic ore seems to have originated from the porphyry, which is more ancient than the granite. The granite breaks through the porphyry dike.

An estimation of tonnage is difficult, owing to the irregular shape of the deposit and the lack of sufficient development. Some have thought that both deposits together may contain 1,500,000 tons.

The Vladimir deposit is a little the more developed of the two. There is a tunnel 100 m. long, and there are several shafts. The ore is magnetite, and ranges from 45 to 53 per cent. of iron. Rocks and minerals like those at Olga outcrop around the deposit.

Chromium, Platinum, Quicksilver, Manganese.

There is a chromium-deposit in the northern part of this Ussuri district, which, conformably with the usual experience, is in serpentine, but no development-work has been done upon it.

The manganese-deposits which appear in connection with limestone, have also not been developed.

Platinum was discovered at the mouth of the Amadyr river many years ago, but probably the alluvial sands are unworkable. More recently, iridium has been found in one of the coast rivers, with the alluvial gold. It lies close to the Manchurian boundary, where I personally examined the occurrence. Quicksilver has been reported in the Amadyr valley.

Coal-Deposits.

There are numerous coal-seams in the Vladivostok district, including Saghalien island. The coal-measures seem to exist all along the Sikh-ot-alin range, which runs parallel to the coast until cut out in its southern end by the sea, making the salient *rias* of the Vladivostok coast. The Sikh-ot-alin is a folded chain. The strata are dislocated, especially the oldest formation, a fact liable to affect also the coal-deposits. There are three distinct coal-formations: the lignites, the Mesozoic and Tertiary coals, and the anthracites.

The lignites are worked about 25 miles from Vladivostok; the production was 10,700 tons in 1910, and about as much annually in several preceding years. The Mesozoic and Tertiary coals are found on the island of Saghalien. There, before the Japanese war, in various places the production was from 25,000 to 30,000 tons a year; the output diminished to 11,500 tons in 1910. The seams are much tilted, in fact almost to the vertical. The most ancient horizon is the upper Cretaceous (Senonian), but the largest seams are in the lower Tertiary, adjoining the Cretaceous. They follow a N–S. line, about 90 miles long. The Miocene and Pliocene formations also contain some coal-seams. Among the best of the Saghalien coals is the product of the Pilevo mine, containing from 70 to 85 per cent. of carbon. Pilevo enjoys, moreover, the rare advantage of a partly-protected harbor, while the whole western coast of Saghalien, with its coal-mines, has no harbor.

The anthracites occur east and west of Vladivostok. The most important mine at present is Su-Tchang, which belongs to the Crown. This mine, about 85 miles from Vladivostok, is connected with the city by railroad. The coal is somewhat poor in quality, and is very brittle. Its calorific value is 25 per cent. lower than that of Cardiff coal. This coal is used by steamers, because there is nothing else to be had, but it is far from satisfactory. The annual production of Su-Tchang is between 120,000 and 200,000 tons. The working-costs are very high, and the mine would probably be abandoned if another deposit of superior quality were available.

New developments are probable at Mangugai, where there is a coal containing 76 per cent. of carbon. Mangugai is 24 miles from Vladivostok, half way by sea, and half way by land. Tests have been made in comparison with Cardiff coal, and the calorific value was nearly the same. The seams dip at a gentle angle, from 16° to 24°, and run N. 43° E., or about parallel with the Sikh-ot-alin range. But little development-work has been done as yet. There are several seams, the best one being from 2 to 3 ft. thick. The Saghalien seams occasionally are 6 ft. thick, with a firm roof and floor. It is hard to believe that so little work has been done when the conditions are so favorable.

As regards consumption, there is a fairly-large market at

Vladivostok. More than 350 steamers arrive annually, each requiring an average of 1,000 tons of coal. The Japanese coals are inferior to the Mangugai coals, and the Chinese coal is also probably inferior. Both are too far away to compete with them. China, with six principal mines, has sold in 1910 more than 4,600,000 tons of coal, as follows:

Kaiping, near Tien-Tsin,	1,174,000 tons.
Fushun (Mukden), near Mukden,	830,000 tons.
Ping-Siang, near Kiao-Tchow,	610,000 tons.
Shan-Tung, near Kiao-Tchow,	483,000 tons.
Lanchow, near Tien Tsin,	360,000 tons.
Pekin-Syndicate, near Tien Tsin,	357,000 tons.

These deposits belong mainly to English and German companies, and would not affect in any way the coal-market of Vladivostok.

Oil-Deposits.

Oil has been struck on the island of Saghalien. The Tertiary formations have been penetrated under a thick cover of post-Pliocene strata which overlie transgressively the oil-formations, supposed to be Miocene.

The oil-strata ought to extend about 170 miles north and south. All the oil locations follow this line along the eastern coast, while coal is on the western coast. The direction runs parallel with the tectonic back-bone of the island, yet the locations of oil are in three different horizons, with no decision as yet as to which is the best one.

According to similar conditions in Japan, the average content of the sands would be between 30 and 50 per cent. of petroleum, perhaps higher. More prospecting-work across the post-Pliocene beds will be required in order to ascertain whether the pools will be commercially productive.

SUBJECT TO REVISION.

[TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS.]

The St. Helens Mining-District.

BY HORACE V. WINCHELL, MINNEAPOLIS, MINN.

(Cleveland Meeting, October, 1912.)

Location.

THE St. Helens mining-district, indicated in sketch-map, Fig. 1, is chiefly in Townships 9 and 10 North, Ranges 5 and 6 East, of the Willamette meridian, in Skamania county, Wash. There is no railroad or even a wagon-road at present connecting it with any base of supplies; access is usually by wagon-road for about 50 miles from the town of Castle Rock to Spirit lake, and from that point by pony-trail from 6 to 10 miles farther.

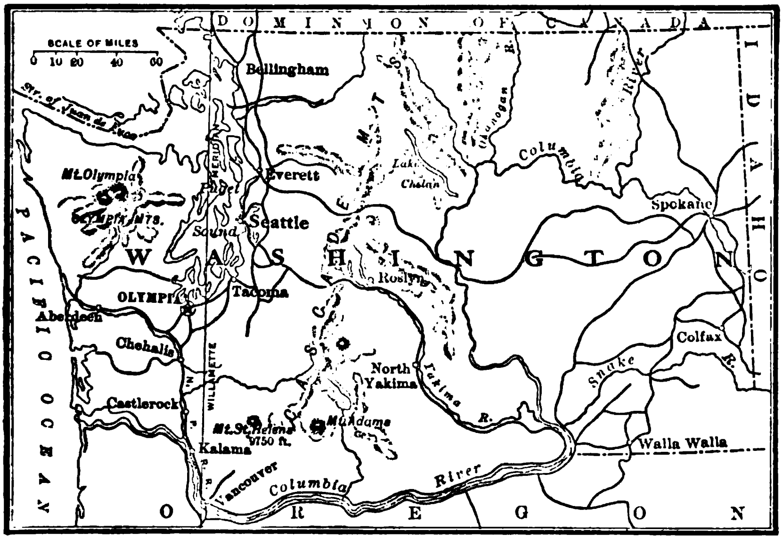

FIG. 1.—SKETCH-MAP OF THE STATE OF WASHINGTON.

History.

Nearly 20 years ago two German farmers, totally inexperienced in prospecting, penetrated this region upon a hunting- and fishing-trip. Their attention was drawn to numerous joints and crevices filled with black, glistening tourmaline, and containing interspersed crystals of pyrite, chalcopyrite, arsenopyrite, specular hematite, magnetite, and occasionally galena and zinc-blende. It seemed to them that they had stumbled upon a veritable Golconda, perhaps the center of mineralization of the entire region. After making as many mining-locations as possible, considering their limited supply of food, they returned to their homes with all the sensations of millionaires, and a determination to come again in the following spring and locate a few more bonanza mines.

From the enthusiasm born of ignorance, and solely through the reports and specimens of these two men, the prospecting-fever spread so rapidly that within three years there were 500 men in the district, and locations were made without number. The proprietors of a German newspaper published in Milwaukee became interested. They furnished the capital; and under the advice of their "expert," who was naturally one of the discoverers of the district, spent many thousands of dollars in prospecting-work. Mining companies of large capitalization were organized, and stock sold broadcast. Thousands of feet of tunnel were driven for the purpose of penetrating and exposing to view the enormous bodies of ores, the existence of which was supposed to be demonstrated by the tourmaline seams so abundant on the surface. The developments were not altogether satisfactory even to the dream-blinded journalists, and they finally concluded to take a moderate price by selling for a few paltry millions to an English syndicate. Then came the first inspection of the district by a man of some knowledge, and a report which suddenly stopped the expenditure of money. But the German prospectors were not to be so lightly divested of the rightful fruits of their discoveries. Others were interested, other companies were incorporated; abandoned claims were relocated and more stock sold, and in the course of time these mining-claims came to have a real value for the timber with which they were covered; and indeed some of it is wonderful. In order to secure this timber, mining-locations were plastered over the country in solid blocks with parallel end-lines and side-lines covering thousands of acres; and in order to keep up the pretensions of a mining-region, more tunnels were driven into the solid rock following the tourmaline-joints.

The total amount of development-work in the district amounts to about 11,000 ft., most of which I examined in the fall of 1911.

Topography and Climate.

The elevation ranges from less than 2,000 ft. above the sea-level to nearly 5,000 ft. The country in general is covered with a heavy growth of forest primeval. The undergrowth is abundant, and consists largely of huckleberry bushes, devil's club, and other shrubs, which frequently make it impossible to see any considerable distance ahead through the forest. The soil and wash covering the solid formation have a considerable depth. There is a mantle of pumice-stone, derived from near-by extinct volcanoes, which in many places has a thickness of several feet. The rocks are generally concealed from observation. The only places where exposures are continuous and clear for any considerable distances are the gorges of the mountain torrents and the tops of mountains, where the rocks project in steep bluffs. Sometimes such outcrops are found along and in the vicinity of Green river, which follows the main axis of depression throughout this district in a N–NW. direction. The snow-falls and rain-falls are evidently heavy, and the result upon the surface of the country-rock is that of a rapid erosion. There has also been in recent times a glacial mantle covering considerable areas, as is shown by the glacial striations upon surfaces of exposed country-rock. The mountain torrents usually flow over bare country-rock which is smooth and precipitous, and frequently marked by joints or slight steps. At other times it is almost as smooth as a toboggan-slide, making what is known in the Alps as a "glissade." These glissades are numerous and afford excellent exposures along a narrow belt of the underlying country-rock.

Geology.

Geologically, the country embraced within this district is comparatively recent. The main mass of the country-rock consists of diorite, a deep-seated eruptive of granitoid texture, consisting chiefly of plagioclase or soda-lime feldspar, and hornblende. With a very few exceptions, to be noted later, this rock is everywhere extremely fresh, comparatively uniform in texture, and seldom contains visible quartz crystals. Towards the northern portion of the district, however, quartz is noticed and the rock there is a quartz-diorite. In this diorite are many secondary or associated minerals. The so-called accessory minerals of marked importance and significance are pyrite, tourmaline, chalcopyrite, specular hematite, magnetite, and epidote. These minerals are in this instance not vein-minerals, but are accessory minerals belonging to the diorite itself.

The other rocks present include andesite and basalt. The andesite is later than the diorite and penetrates it in the shape of dikes. There are not so many or so largely developed accessory minerals in the andesite as in the diorite.

The basalt is found only in the NW. corner of the district. It is a dark-colored basic eruptive rock, fine grained, and perhaps the product of a more recent igneous activity which has broken out since the cooling of the diorite. No accessory minerals of any considerable quantity were noted in the basalt. Basalt and andesite tuff are neither of them formations of such origin, age, or history as to be particularly favorable for the deposition of ore.

Joints are fairly abundant, although over many portions of the district the interspaces are from 10 to 20 ft. broad.

Attractive to the Prospector.

Situated in a region of good climate, abundant water, and ample shelter, with structural material upon either hand, affording fuel at his door, the St. Helens district a few years ago, though generally unknown and mountainous, must have been decidedly stimulating to the imagination of the average prospector.

Upon first going to the St. Helens district he would observe in the water-courses and in the outcroppings of rock those indications of mineral which the surface affords, and nowhere any excavation to disclose what the underlying rock contains. The rocks present smooth, bare exposures, in which any structural features are plainly evident. There has been a slight surficial iron-staining in the vicinity of joints and crevices. There are upon the surface, scattered throughout the rock and associated with joints and other crevices, pyrite, chalcopyrite, specular hematite, magnetite, tourmaline, epidote, and occasionally arsenopyrite, galena, and sphalerite. All of these minerals in disseminated particles or in thin bands of segregation are of frequent occurrence.

Another favorable feature in connection with this district is the possibility of exploring to considerable depth by driving tunnels of no great length into the mountain-side at almost any point without the necessity of expensive machinery; and without having at his command financial resources of any considerable importance, the prospector is able by his own efforts to investigate the underground conditions and the significance and value of those indications which are found upon the surface. To the experienced prospector, the indications will not seem important or attractive, even upon its surface. To the inexperienced prospector, who is unable to distinguish specular hematite from galena, who cannot tell gray copper from a mixture of pyrite and arsenopyrite, who does not know the difference between chalcopyrite and tarnished massive pyrite, there would probably be an inducement to perform some work of exploration. He could even procure, as the result of hundreds of dollars' worth of work and weeks and months of labor, an occasional specimen for exhibit and even for assay, a specimen which might attract the attention of men with money, who themselves were unacquainted with mining and geology,—specimens which, lying in a broker's office, might sell stock to the uninformed. The prospector of experience would notice that there are no large placer-deposits. This is of peculiar significance and importance. In almost every region and mining-district where sulphide ore-deposits exist in large quantity, and where there has been superficial oxidation, decomposition of the rocks, and accumulation in the valleys of the products of erosion, the insoluble gold and silver of the ores have been accumulated in the form of metallic products in sufficient quantity to make placer-mining profitable. The St. Helens district is conspicuously devoid of such deposits. The country is a region of young topography. The valleys are new. They are not filled with any considerable quantity of gravel. Throughout the entire district which came under my observation, there is no indication of an accumulation of mineralized material or of free gold in placers.

Another thing which would strike the attention of the experienced prospector is the universal absence of quartz in vein outcroppings and in float. In other districts where large quartz veins occur, they are usually, although not always, harder than the country-rock in which they are inclosed. As a result, during the progress of erosion, the quartz veins project above the surface of the surrounding country and finally are worn down in fragments, and "float quartz" has been the means of guiding by far the larger number of prospectors to the veins which they are seeking and from which the quartz has been derived. No such quartz has been found in this district.

Still another fact would impress itself upon the experienced prospector, and that is the entire absence of gossan or iron-hat. Everywhere sulphide minerals occur in the rocks at the surface and in the joints within a few feet of the surface. So thorough has been glacial scouring, so rapid has been erosion, that oxidation has hardly begun.

The iron-hat, or gossan, frequently referred to as an important and valuable indication of ores in depth, is a zone of decayed and oxidized rock which covers the upper portion of veins and country-rock alike. The country-rock may not be rotted and oxidized to such a depth as the veins, but in any district where there is extensive gossan, the country-rock also is deeply rotted and oxidized. Now, this means that the sulphide minerals have been almost entirely removed. No iron-hat exists where fresh, bright, glittering pyrite and chalcopyrite minerals occur within a few inches of the surface. No gossan exists where the ferro-magnesian minerals of the eruptive rocks are perfectly fresh and sharp and bright, and where the feldspar shows no trace of decay. Furthermore, if the prospector were to enter this district to-day and see what is to be seen underground in many tunnels, he would discover no secondary enrichment. The natural, universal, and inevitable result of the oxidation of sulphide minerals, such as pyrite and chalcopyrite, in a vein or mass of country-rock, and its downward percolation into the rocks below, is the formation of secondary sulphide minerals. The principal secondary sulphide of copper is copper-glance, or chalcocite. This mineral is universally found where pyrite and chalcopyrite have been dissolved and carried downward through and along the unoxidized minerals below. The only possible exception to this statement just made is in a limestone-region where the carbonate of lime is so soluble and occurs in such quantity that it may precipitate from the sulphate solution all of the copper as carbonates. This seldom occurs, and even in our largest limestone copper-camp, at Bisbee, Ariz., although the carbonates are abundant immediately beneath the surface, the larger portion of the ore is secondary "glance."

Another thing which would impress the experienced prospector at the outset, and the average prospector if he saw the district to-day, is the universal fact that there is no improvement or increase of mineralization as depth is gained in the prospecting-work already performed. On the contrary, there is a general rapid diminution of alteration of the country-rock. There is no ore in commercial quantity. There are only specimens, and the specimens are obtained by handling and extracting many tons of rock for one ton of specimens. The last phenommenon which would impress itself upon the notice of the intelligent prospector is the absence of softening and alteration of the country-rock adjacent to the joints and seams. Wherever vein-mineralization has been extensive and important, and has effected the deposition of valuable ore-deposits, the adjacent rock is usually widely decayed and altered, not by oxidation, not by weathering, not by any surface-action, but through the solvent action of the solutions which deposited the ore in the veins. In Butte, for example, the granite and porphyritic rock inclosing the veins is altered, decayed, and mineralized with pyrite for hundreds of feet away from the main channels of mineralization. No such alteration occurs here. On the contrary, the country-rock is bright, fresh, clean, hard, and ringing within a few inches of the joints.

A prospector who has some knowledge of mineralogy and of the origin of minerals would be impressed by one or two additional facts. The joints are associated with and enlarged by deposits of epidote, tourmaline, and magnetite. Tourmaline is a mineral formed from vapors. It is what is known as a fumarolic or pneumatolytic mineral. It is not formed by ordinary vein-waters. It occurs in the vicinity of and along joints, and it was deposited before the rocks were cold, and possibly before they were sufficiently cool to enable vein-solutions to exist. The chalcopyrite in this entire district is so universally associated with this mineral and with specular hematite, magnetite, epidote, and the usual concomitant sulphide minerals, such as arsenopyrite, sphalerite, and galena, that there can be no question whatever as to the date and manner of deposition; the origin of these sulphide minerals in the joints and associated with the joints is pneumatolytic. Pneumatolytic copper-tourmaline ores are rarely deposited in sufficient quantity to make important copper-mines. They represent the initial stages of mineralization, and where they still occur in their original character and position there is seldom anything of value.

There has been one other form of deposit of copper and iron sulphides in the St. Helens district: that is, purely magmatic segregations of sulphide from a molten mass, just as matte separates from the slag in the slag-pot. This form of deposit is exemplified at the Polar Star, where there are the usual blotchy and scattering bunches of sulphide minerals in the solid unaltered country-rock.

The prospector of judgment and experience must be a man of some knowledge of geology and mineralogy. He must have some conception as to the conditions under which ores can be worked profitably, and how much ore it takes to make a mine. Such a prospector going into this district to-day could not fail to see on every hand evidences of disappointing results and of general barrenness.

In the preparation of the above description, from 25 to 30 different claims and groups of claims were examined. The amount of development varied from several hundred feet of tunnel and drifts to small open pits. The maximum showing on which this work was based was a brecciated zone 6 ft. wide; the minimum was a mere joint-crack. In the commonest experience there had been revealed from 1 to 2 ft. of brecciated or sheeted rock, with small but practically barren veinlets. At the best, after hand-sorting many tons of the run of the mine, comparatively few pounds of low-grade copper sulphides were obtained and other ores are almost entirely absent.

SUBJECT TO REVISION.

[TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS.]

A Titaniferous Iron-Ore Deposit in Boulder County, Colo.

BY E. P. JENNINGS, SALT LAKE CITY, UTAH.

(Cleveland Meeting, October, 1912.)

LARGE deposits of titaniferous iron-ore occur at Caribou, an old silver-mining camp in Boulder county, Colo., 17 miles west by south of Boulder, and a few miles northwest of the tungsten-mines. Professor Regis Chauvenet[1] published a part analysis of the ore many years ago, but gave no description of the wall-rock.

Caribou hill, which contains both the silver-bearing veins and the iron-ore deposits, is a low dome of gabbro surrounded by the Archæan schists and gneisses of the Front range of the Rocky mountains, the peaks of which, a few miles west of Caribou, attain an elevation exceeding 12,000 ft. The gabbro stock is roughly elliptical in outline, with approximate diameters of 1 and 3 miles. The crest of the dome is 10,500 ft. above sea-level, and the town and ore-deposits are 500 ft. lower. Outcrops of the iron-ore occur in the town and extend NW. for 2,000 ft., where they disappear under the swampy meadows of Caribou park. Portions of the deposit are covered by drift, but one continuous outcrop is 900 ft. long and from 200 to 250 ft. wide. A second deposit is 0.25 mile south of the town along the eastern slope of the hill. It is not as well defined as the north bed, but is approximately 1,000 ft. long and from 100 to 200 ft. wide. The easily disintegrated ore does not form prominent outcrops, but the outlines of the deposits are readily traced by the large amount of ore fragments that cover the ground. Trenches and pits sunk on the outcrop expose a roughly banded magnetic iron-ore, with pyroxene as the principal gangue-mineral.

The Gabbro.

This rock was classified in the field as a gabbro from the presence of pyroxene and striated feldspars, and for convenience the field-name will be retained.

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[1] Chauvenet, Regis, *Biennial Report of State School of Mines*, Golden, Colo., p. 16 (1886).

Megascopic Characters.

The rock is light to dark gray, even-grained and granular. It weathers light brown. It is medium-coarse in texture in the central portions of the stock, and fine-grained near the contacts with the gneiss, but there is no sharp line of demarcation between the two varieties. The silver-veins occur in the coarse-grained variety, and the iron-ores in the finer-textured rock.

Under a lens both varieties show abundant striated feldspars, white orthoclase, dark-green pyroxene, iron-ores, and brown biotite. The latter mineral is abundant in the coarse-grained rock, but scanty or entirely absent in the fine-grained phase.

Microscopic Characters.

Thin sections (Figs. 1 and 2) show the following minerals: Iron-ores, apatite, pyroxene, biotite, plagioclase, and orthoclase. The plagioclase is in clear lath-shaped crystals that in the fine-grained rock average from 1 to 2 mm. in length, twinning after the albite and Carlsbad laws and containing scanty apatite inclusions. In the coarse-grained variety, the feldspars are from 3 to 5 mm. long, and the other minerals proportionately larger. Determinations by the statistical method and extinction angles in zone at right angles to 010, where albite and Carlsbad twinning are combined, give: Ab 1.5, An 1 to Ab 1, An 1. These ratios make the plagioclase a variety in the andesine series on the border with labradorite. Orthoclase, which is the last mineral to separate, occurs in irregular forms, filling spaces between the other minerals; it is clear to milky and contains abundant inclusions of ore and pyroxene. Colorless to pale yellowish green, non-pleochroic grains and irregular masses of pyroxene form about 10 per cent. of the rock. Maximum extinction angles of 39° to 40° from C. axis place the mineral near diopside. Biotite in shreds and plates, closely associated with the diopside, forms about 3 per cent. of the coarse-grained rock, but is often absent in the fine-grained variety. Stout prisms and hexagonal grains of apatite are rather abundant as inclusions in all the other minerals. Iron-ore is abundant as inclusions in the diopside and orthoclase, but is absent from the plagioclase.

The order of crystallization is: apatite, plagioclase, ores, biotite, diopside, orthoclase.

The analyses given in Table I. were made by the methods recommended by Hillebrand[2] of fresh material taken from shafts and tunnels:

TABLE I.—*Analyses of the Gabbro.*

	I. Coarse-Grained Rock from Shafts Near Center of the Gabbro Stock, Contains Small Amount of Biotite.	II. Fine-Grained Rock from Tunnel Near Ore-Bed, Northern Edge of Gabbro.
	Per Cent.	Per Cent.
SiO_2	51.43	52.10
Al_2O_3	19.35	19.35
Fe_2O_3	5.32	4.63
FeO	4.70	4.16
MgO	3.05	3.19
CaO	7.09	7.15
Na_2O	3.54	3.39
K_2O	3.52	3.77
H_2O + 105°	0.35	0.13
H_2O — 105°	0.08	0.06
TiO_2	0.85	0.94
P_2O_5	0.60	0.71
MnO	Tr.	0.10
	99.88	99.68

The analyses prove the rock to be a gabbro closely related to the syenites. The high potash percentage with the consequent orthoclase and biotite remind one of the gabbros at Duluth, Minn., and others in the Cortlandt series near Peekskill, N. Y. When recast according to the methods of the quantitative system for the classification of igneous rocks, the following norms or theoretical percentages of different components are obtained:

	I.		II.	
Orthoclase	20.57	76.60	22.24	77.19
Albite	29.34		28.82	
Anorthite	26.69		26.13	
Diopside	4.23	22.63	4.20	22.20
Hypersthene	6.32		7.02	
Olivine	1.56		0.90	
Magnetite	7.66		6.73	
Ilmenite	1.52		1.67	
Apatite	1.34		1.68	
H_2O	0.43		0.19	
	99.68		99.59	

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[2] Hillebrand, W. F., The Analysis of Silicate and Carbonate Rocks, *Bulletin No.* 305, *U. S. Geological Survey* (1907).

I. II.

$$\frac{\text{Sal}}{\text{Fem}} = \frac{76.60}{22.63} = 3.38 = \text{Class II.} \qquad \frac{77.19}{22.2} = 3.47 = \text{Class II.}$$

$$\frac{F}{Q} = \frac{76.60}{0} = > \frac{7}{1} = \text{Order 5} \qquad \frac{77.19}{0} = > \frac{7}{1} = \text{Order 5}$$

$$\frac{Na_2O'\ K_2O}{CaO'} = \frac{93}{96} = 0.96 = \text{Rang 3} \qquad \frac{95}{94} = 1.01 = \text{Rang 3}$$

$$\frac{K_2O'}{Na_2O'} = \frac{37}{56} = 0.66 = \text{Subrang 3} \qquad \frac{40}{55} = 0.72 = \text{Subrang 3}$$

The magmatic symbol for both rocks is: II. 5. 3.3, or Shoshonose, by the chemical classification. I would classify the rock as an orthoclase-biotite gabbro in preference to calling it a monzonite, on account of the predominance of the plagioclase.

The Iron-Ores.

The principal ore-deposit is a pyroxene-magnetite-biotite dike, 250 ft. wide, in which the minerals are very unequally distributed, the magnetite being largely concentrated in the central portions, while the biotite is confined to a zone near the walls.

The order of distribution from the walls to the center is: 25 ft. of biotite-pyroxene rock (Figs. 3 and 4), with accessory plagioclase, magnetite, and orthoclase; 50 ft. of pyroxene rock, with little magnetite and biotite and no feldspar; the remaining 50 ft. to the center of the dike is a low-grade iron-ore, averaging from 30 to 35 per cent. of iron, and containing about equal amounts of magnetite and pyroxene. Samples can be selected from this central zone that are nearly pure magnetite, but the rich bands are but a few inches wide and are difficult to separate from the leaner ore. There is no sharp division between the different zones, but a gradual merging of one into the other.

The ore, which is strongly magnetic, is an intimate mixture of titaniferous magnetite and pyroxene, with accessory spinel, apatite, and biotite. Its texture varies from a coarse-grained granular aggregate to one extremely fine-grained, that has a slaty structure resembling the slate ores of Lake Superior.

Thin sections (Figs. 5, 6, 7, and 8) show abundant stout prisms and grains of colorless to slightly yellowish green pyroxene, from 0.1 to 5 mm. in length, often in well-defined crystals, distributed quite uniformly in the magnetite ground-mass.

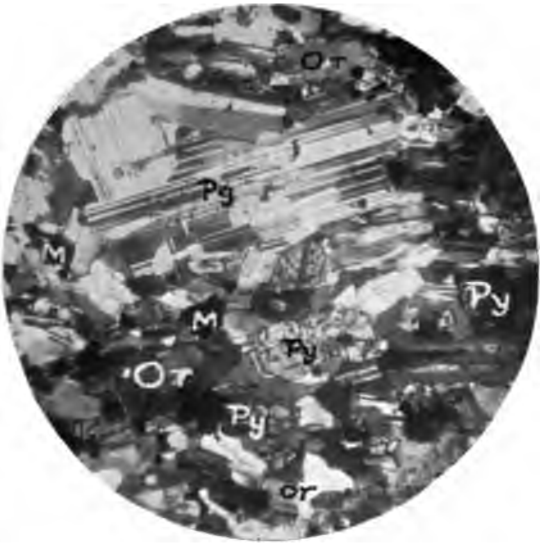

FIG. 1.—FINE-GRAINED GABBRO, 24 DIAMETERS; POLARIZED LIGHT; NICOLS CROSSED. ACTUAL FIELD, 3 MM. PLAGIOCLASE (PG), ORTHOCLASE (OR), PYROXENE (PY), MAGNETITE (M).

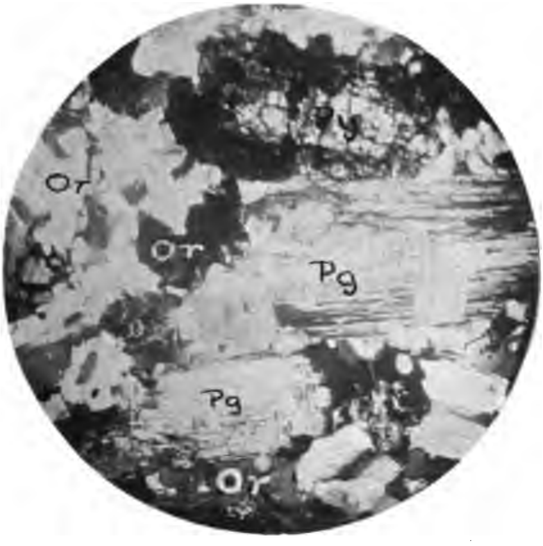

FIG. 2.—COARSE-GRAINED GABBRO, 18 DIAMETERS; POLARIZED LIGHT; NICOLS CROSSED. ACTUAL FIELD, 4 MM.

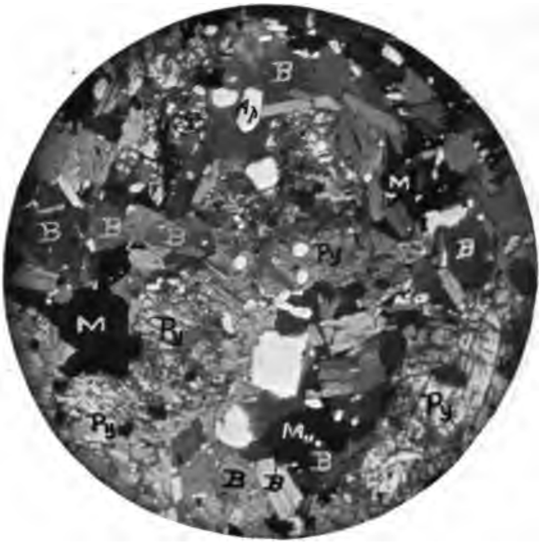

Fig. 3.—Biotite-Pyroxene Zone of Dike, 9 Diameters; Ordinary Light. Actual Field, 8 mm. Apatite (Ap), Biotite (B), Pyroxene (Py), Magnetite (M).

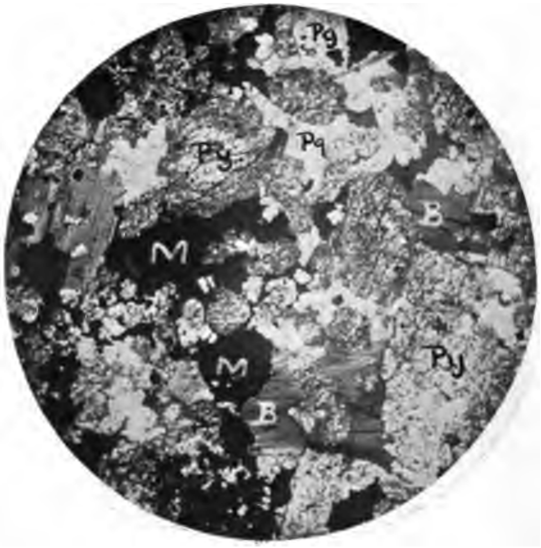

Fig. 4.—Outer Edge of Dike, 9 Diameters; Ordinary Light. Actual Field, 8 mm. Plagioclase (Pg) is Last Mineral to Separate.

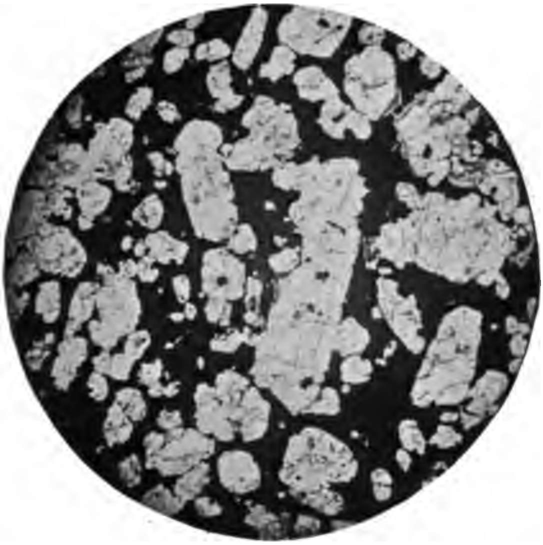

Fig. 5.—Ore of Average Composition, 9 Diameters; Ordinary Light. Actual Field, 8 mm. The Light-Colored Areas are Pyroxene.

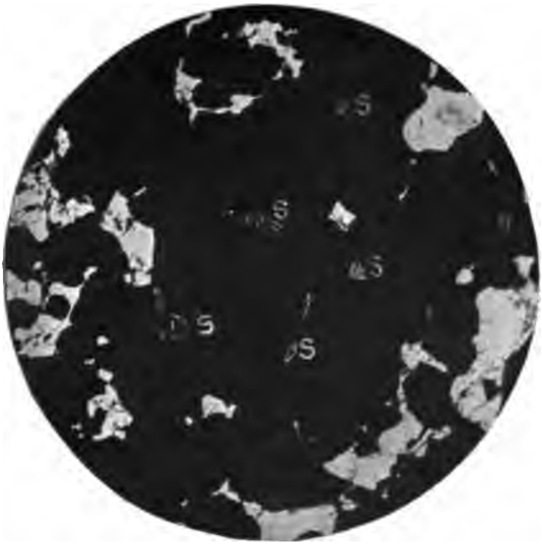

Fig. 6.—Ore from Central Zone of Dike, 18 Diameters; Ordinary Light. Magnetite, Diopside, Spinel (S).

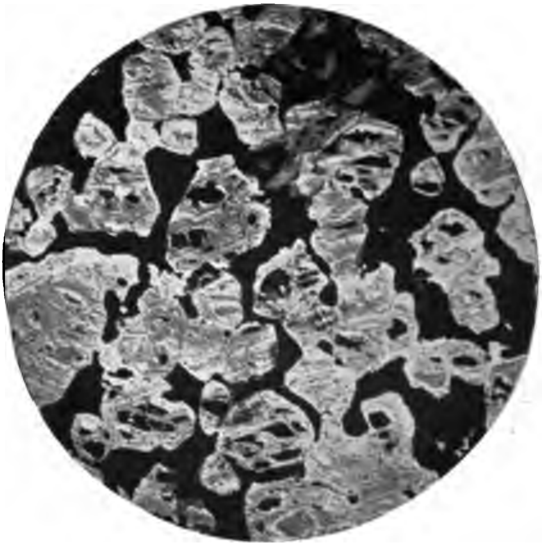

FIG. 7.—ORE FROM SURFACE, DIOPSIDE ALTERED TO SERPENTINE, 24 DIAMETERS; POLARIZED LIGHT; CROSSED NICOLS. ACTUAL FIELD, 3 MM.

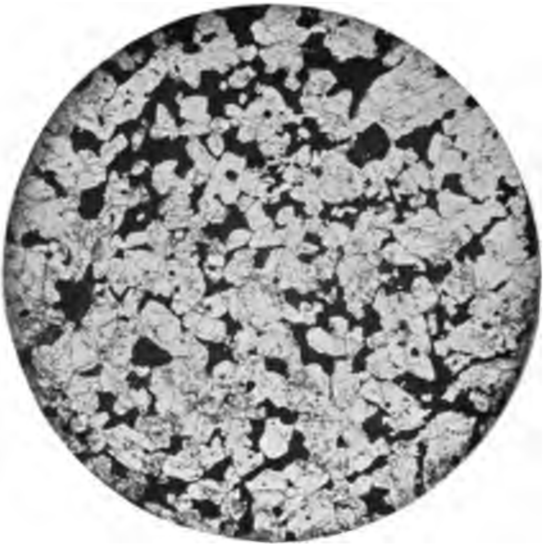

FIG. 8.—LEAN ORE, PYROXENE ZONE OF DIKE, 9 DIAMETERS; ORDINARY LIGHT. ACTUAL FIELD, 8 MM.

They contain a few inclusions of biotite and magnetite (or ilmenite) and needles of apatite; are non-pleochroic, and have a maximum extinction angle of 40°, which places the mineral near diopside. A part alteration of the diopside to serpentine is common in the surface-ore.

Dark-green isotropic crystals of quadratic outline and high refractive index, seldom over 0.1 mm. in diameter, occurring as rather abundant inclusions in the magnetite, were identified as spinel. In 24 sections examined, 12 contained spinel. It appears to be confined to the richer ores, and is always absent from those sections in which the diopside crystallized ahead of the magnetite. Apatite in stout prisms and hexagonal grains occurs as inclusions in both the magnetite and pyroxene, but is relatively more abundant in the lean ore in which the pyroxene predominates. The order of crystallization in the lean ore is: apatite, biotite, diopside, magnetite. In ore containing about 70 per cent. of magnetite, the diopside and magnetite crystallized simultaneously, but apatite and spinel were the first to separate. In the rich ore the order of crystallization is: apatite, spinel, magnetite, diopside. The composition of the eutectic is probably near 70 per cent. of magnetite (including spinel) and 30 per cent. of diopside.

Attempts were made to separate ilmenite from magnetite by means of an electro-magnet without success, as were also experiments in etching polished surfaces with dilute HCl.

The inference that the ilmenite is very finely divided and intimately mixed with the magnetite, is strengthened by the easy and complete solubility of the iron oxides in hot dilute HCl, that is in marked contrast to the difficultly soluble ores of Minnesota and Canada, in which the ilmenite occurs in grains of appreciable size. The ore is strongly magnetic, and many of the surface-boulders are natural load-stones.

Analyses of the Ore.

The following method was used in analyzing the ore: 5 g. of ore was treated with hot HCl, which dissolved all the magnetite and ilmenite; the solution was filtered from the insoluble residue; after which the filtrate was analyzed by the methods of the chemists of the U. S. Steel Corporation. The insoluble residue, consisting of diopside, spinel, and small amounts of TiO_2, was fused with $Na_2S_2O_7$, so as to decompose the spinel and TiO_2, but not the diopside; the melt was dissolved in hot dilute HCl and filtered from the insoluble diopside; the TiO_2 was separated and was added to the TiO_2 obtained in the portion soluble in HCl. The SiO_2, Al_2O_3, Fe_2O_3, and MgO were determined, and the results obtained were taken as the composition of the spinel. The diopside was analyzed by the ordinary methods for silicate minerals. Direct determinations of FeO were made in the portion soluble in HCl by titration with bichromate.

There may be some question as to the accuracy of the method of separating the minerals, but the presence of only sufficient lime to combine with P_2O_5, and its entire absence in the spinel-analyses, is evidence that the diopside is not attacked, either by the HCl or $Na_2S_2O_7$.

The Al_2O_3, SiO_2, and MgO found in the portion soluble in HCl are believed to be parts of the magnetite molecule, and not derived from either the spinel or the diopside.

Table II. gives the analysis of medium-grade ore from the north deposit.

TABLE II.—*Composition of Medium-Grade Ore From North Deposit.*

		Per Cent.		
Soluble in HCl . . .	FeO with TiO_2	4.02	Ilmenite,	8.50
	TiO_2	4.48		
	FeO	20.10	Magnetite,	67.25
	Fe_2O_3	44.72		
	SiO_2	0.31		
	Al_2O_3	0.81		
	MgO	1.31		
	CaO	0.10	Apatite,	0.18
	P_2O_5	0.08		
Portion decomposed by $Na_2S_2O_7$	SiO_2	0.20	Spinel,	6.70
	Al_2O_3.	4.29		
	FeO	0.53		
	MgO	1.68		
By fusion with Na_2CO_3 .	SiO_2	9.09	Diopside,	16.89
	Al_2O_3	1.17		
	FeO	0.66		
	CaO	3.65		
	MgO	2.17		
	Na_2O	0.15		
	K_2O	Tr.		
	H_2O + 105°	0.51		0.51
		100.03		100.03

	I. Spinel, Calculated from Ore-Analysis.	II. Diopside, Calculated from Ore-Analysis.	III. Spinel, Direct Analysis of Separated Material.	IV. Diopside, Direct Analysis of Separated Material.
	Per Cent.	Per Cent.	Per Cent.	Per Cent.
SiO_2	2.98	53.82	2.96	54.24
Al_2O_3	64.03	6.93	64.28	7.03
FeO	7.91	3.90	7.67	3.97
CaO	......	21.62	......	21.41
MgO	25.08	12.85	24.46	12.65
Na_2O	.....	0.88	......	0.87
K_2O	......	Tr.	......	Tr.
	100.00	100.00	99.37	100.17

The spinel is the iron-magnesian variety, pleonaste. The diopside is high in Al_2O_3, but its extinction angle of 40° is that of diopside. Many part analyses were made to determine the ratio of TiO_2 to Fe, resulting in an average of 1 to 10.7, the highest being 1 to 11.4, and the lowest 1 to 8.5. Professor Chauvenet's analysis differs greatly from these results, being TiO_2, 36.76; and Fe, 36.93 per cent.

These interesting ore-deposits have little or no economic importance, but are excellent examples of iron-ores of igneous origin. They may be considered either as true dikes, or as segregations in place from the wall-rocks. The first supposition is the most probable, for the following reasons: In form the deposits are typical dikes of great length and uniform strike, in marked contrast with the irregular outline of most segregated ore-bodies; the wall-rock contacts are sharp and well defined; there is no local impoverishment of the gabbro in ferro-magnesian minerals near the dike, the ratio of these to the feldspars being essentially the same in rock near the dike as in rocks from points 1,000 ft. away, indicating no concentration of basic minerals from the wall-rock.

Similar occurrences have been described as dikes by Kemp,[3] in New York State, and Ball,[4] in Wyoming. Also, Vogt[5] classes certain Norwegian deposits as true dikes.

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[3] Kemp, J. F., Titaniferous Iron Ores of the Adirondacks, *Nineteenth Annual Report, U. S. Geological Survey*, part iii., p. 414 (1897–98).

[4] Ball, Sydney H., *Bulletin* No. 315, *U. S. Geological Survey*, p. 206 (1907).

[5] Vogt, J. H. L., *Om dannelse af de vigtigste i Norge og Sverige representerede grupper af jern-Malmforekomster*, p. 14 (Christiania, 1892).

he inference is that the origin of the dike was in a deep-
ted segregation of ferro-magnesian minerals and magnetite,
to a differentiation from the gabbro magma, and that its
position is essentially the same as that of the parent basin.
he later rearrangement of the minerals of the dike, due to
v cooling under a great mass of superimposed rock, has
tly concentrated the magnetite near the center and the bio-
along the walls.

he order of the deposition of the minerals corresponds
ely with the order of crystallization observed in thin sec-
s of the ore, the biotite being among the first minerals to
arate, and magnetite with the last in each case.

he original dike-magma contained an excess of Al_2O_3 and
O above that required to form pyroxene and biotite; this
ess separated as spinel, excepting small amounts that passed
the magnetite molecule.

livine, a common constituent of gabbro and titaniferous
s, is notably absent, though not from any lack of MgO in
magma.

SUBJECT TO REVISION.

[TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS.]

Clinton Iron-Ore Deposits in Kentucky and Tennessee.

BY S. WHINERY, NEW YORK, N. Y.

(Cleveland Meeting, October, 1912.)

I AM indebted to L. E. Bryant, of Danville, Ky., President of the Virginia Mining Co., operating coal-mines in Scott county, Tenn., for the following information relating to the existence of the Clinton iron-ore under the NW. slope of the Cumberland Plateau in Kentucky and Tennessee.

During the past 15 years quite an important oil-field has been developed in a region having Wayne county, Ky., as its center, and a large number of wells have been drilled in that and adjacent territory in the search for oil. Many of these wells have been carried to considerable depths below the recognized oil-bearing horizon, and one of them was drilled by Mr. Bryant at Pine Knot, a station on the Cincinnati, New Orleans & Texas Pacific railroad, a few miles north of the point where that road crosses the Kentucky-Tennessee State-line. The elevation of the surface at the crossing is about 1,400 ft. above sea-level. The railroad is located on the dividing ridge between the Cumberland river and Big South Fork. At a depth of 1,720 ft. below the surface the drill reached and passed through a formation which the drillers called "red rock," but which Mr. Bryant classified as Clinton iron-ore, as its geological position corresponded to that formation. The ore at this point was 6 ft. thick and appeared to be of uniform quality throughout. An analysis of the drillings gave: Fe, 33; Mn, 0.25; SiO_2, 3; Al_2O_3, 2; CaO, 13; MgO, 7; P_2O_5, 0.333 per cent.; S, trace.

This analysis seems to confirm the belief that the ore is Clinton, and that it is of a quality equal to the average of the deep or unbleached Clinton ores in Tennessee and Alabama.

This discovery led Mr. Bryant to make further examinations and inquiries, and he found that the stratum called "red rock" by the drillers had been passed through and noted in the great majority of the wells carried down to the same geological horizon, over quite a wide area in and around the oil-region. Its geological position was readily fixed in these wells, as it was noted as being from 50 to 100 ft. below the Devonian black shale, which is very persistent in the region, is universally recognized and referred to by the well-drillers. The thickness of the "red rock" was not usually recorded, but was stated by the drillers to be from 3 to 6 ft.

In a number of wells reaching the horizon of the ore its occurrence was not noted. This is not surprising in view of the well-known lenticular formation of the Clinton ore-deposits in the South.

These facts seem to indicate that rich and comparatively thick deposits of Clinton ore underlie the Cumberland Plateau in this region and extend under its northern slope. If so, it will, undoubtedly, in time, become commercially very valuable, particularly as the whole region is also underlain by three or four workable beds of coal, at least two of which are good coking-coal, so that both ore and coal could be mined from the same or adjacent shafts. The coal outcrops above the surface of the larger streams and is being mined in large quantities by drifts in the region. Shafts sunk along the streams in the vicinity would reach the iron-ore at a depth of about 1,000 feet.

SUBJECT TO REVISION.

[TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS.]

The Ta-yeh Iron-Ore Deposits, Hu-pei Province, China.

BY C. M. WELD, LOW MOOR, VA.

(Cleveland Meeting, October, 1912.)

In the course of my professional work in China during the fall of 1907, I had an opportunity to visit the iron-ore mines at Ta-yeh in Hu-pei province (long. 114° 50′ E., lat. 30° 20′ N.). It occurs to me that a rescript and discussion of my notes taken at that time should be of interest, particularly as supplementing the exceedingly interesting recent paper of Thomas T. Read on The Mineral Production and Resources of China.[1]

Ta-yeh is situated near the right bank of the Yangtze river. The mines are at an average distance of about 16 miles from the river-port Shi-hui-yao, which in turn is some 70 miles S–E. of Hankow. The ores are shipped to the works of the Han-Yeh-P'ing Iron & Coal Co. at Hanyang, as also very largely to the Japanese government iron- and steel-works at Wakamatsu, Japan. Smaller quantities have been exported to the Pacific coast of the United States.

These ore-deposits have been briefly noticed in a number of publications, but, so far as I know, Mr. Read's description, in his chapter on the iron-ores of the Chinese Empire,[2] contains the first attempt to define their geological character. In his more recent paper Mr. Read follows very closely the text of his earlier description. To quote briefly from the recent paper, he says: "The iron-ores at Ta-yeh . . . . lie along the contact between a marble and an intrusive body of a dark-gray syenitic rock." He refers to a figure which exhibits in plan, but not in section, the relative positions of the syenite, ore-bodies, and limestone. The contact is seen to trend approximately E–W., with the syenite on the north and the limestone on the south. "The syenite lies to the north of the marble, and a few miles farther north the marble again appears, iron-ore again being present along the contact." No lithological description of the syenite is given. The limestone is discussed as probably of Carboniferous age, and the suggestion is advanced that "this limestone is a horizon for iron-ores in the Yangtze valley, as they are associated with it in notable amount near Nanking." Again: "In Kiang-su province iron-ores are widely distributed, occurring chiefly in the region about Nanking in association with the limestone previously mentioned." And in the following paragraph: "In the province of An-hwei the geological conditions are similar to those in Kiang-su and Hu-pei." In describing the ore he says: "The ore is a good quality of hematite [here follow analyses] . . . . At one place it is slightly magnetic, apparently having been partly reduced to the magnetic oxide by the action of reducing solutions, which have deposited small amounts of copper- and iron-sulphides along the foot-wall."

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[1] *Bulletin* No. 63, March, 1912, pp. 293 to 343.

[2] *The Iron Ore Resources of the World*, vol. ii., p. 913 (Stockholm 1910).

A. J. Seltzer[3] has described the Ta-yeh deposit as "a contact deposit, the contact on the north wall being a compact, close-grained diorite, the south contact being a pure, almost white, marbly limestone." He presents analyses for three kinds of ore, namely, hematite, magnetite, and manganiferous brown ore.

Another reference which I have before me[4] describes the Ta-yeh ores as including magnetite, hematite, and manganiferous brown ores, in a steeply-dipping vein. "On the section of the deposit where lower phosphorus ores are found, the vein is in contact with almost pure limestone in very large quantities."

My visit to the Ta-yeh mines was only of a few hours' duration; my examination was therefore necessarily superficial, and confined to a single one of the several ore-bodies which have been developed. I gathered no samples or specimens, and have therefore to rely wholly upon macroscopic inspection, made at the time, of the ores and associated rocks and minerals, having neither analyses nor thin sections for further detailed study.

Turning to my notes, I find the ore-bodies described as occurring along an approximately E–W. contact between hornblende-granite and limestone, the former lying north and the latter south of the contact.

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[3] Iron and Steel Works at Hanyang, Hupe, China, *Engineering and Mining Journal*, vol. lxxxix., No. 24, p. 1232 (June 11, 1910).

[4] The Iron Industry of China, *Iron Age*, vol. lxxxi., No. 19, p. 1436 (May 7, 1908).

The granite is a granitoid rock consisting essentially of quartz, potash-feldspar, and hornblende. While ordinarily holocrystalline to the naked eye, some fine-grained and porphyritic facies appear at the contact. The general color-effect on a slightly weathered surface is light gray, peppered with black.

The limestone is a light-gray smooth-textured rock, changing to a coarsely crystalline and often pure white marble at the contact. Seltzer[5] states that this stone carries 96 per cent. of $CaCO_3$. At least it appears to be sufficiently pure to yield an acceptable flux for the Hanyang blast-furnaces, and is quarried for that purpose at a point about 600 yd. south of the contact. Contact-minerals in the limestone appear to be rare; in fact, none were observed except where occasional areas were seen to be abundantly impregnated with minute resinous-brown garnets, frequently arranged in bands. It should be noted, however, that my remarks apply to a section where the granite and the limestone are separated by about 200 ft. of iron-ore; no actual contact between the two came under my observation.

In structure the limestone exhibits a great anticline in the country south of the contact, its axis trending approximately ENE–WSW. The northward-dipping leg of this anticline is bent sharply back upon itself as it approaches the contact. The granite occupies, so far as I could learn, an elliptical area, from 6 to 10 miles E–W. by from 2 to 3 miles N–S. Within this area the granite peaks stand out in somewhat bolder relief than the surrounding limestone country. I have already quoted Read to the effect that the limestone reappears to the north. On the east limestone is seen along the right bank of the Yangtze river, except where overlain by red beds.

Before turning to the ore-bodies it will be convenient to refer briefly to my figures, which have been reproduced from the original sketches included in my notes. Fig. 1 represents the district as a whole, and shows the general position of the mines with reference to the Yangtze river. Fig. 2 represents the western portion of Fig. 1 on an enlarged scale, while Fig. 3 is a geological cross-section taken on a N–S. line and looking

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[5] *Op. cit.*

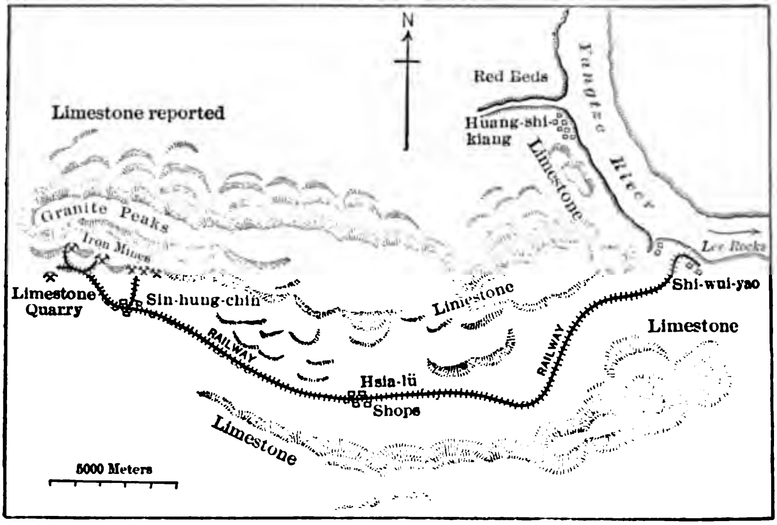

FIG. 1.—SKETCH-MAP SHOWING POSITION OF THE TA-YEH IRON-MINES, HU-PEI PROVINCE, CHINA.

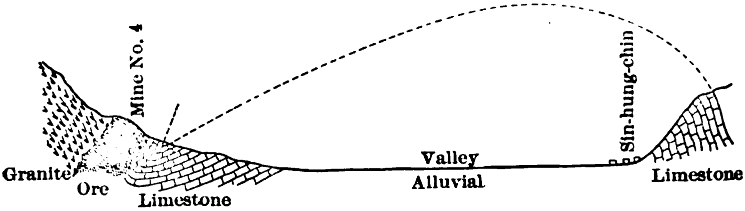

FIG. 2.—GEOLOGICAL CROSS-SECTION THROUGH MINE NO. 4 AND SIN-HUNG-CHIN, LOOKING EAST.

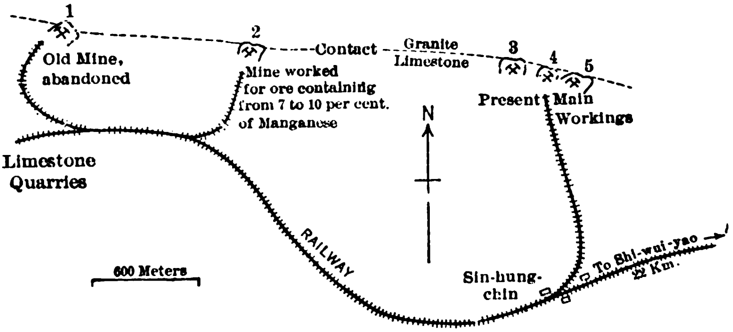

FIG. 3.—SKETCH-MAP OF THE TA-YEH IRON-MINES, HU-PEI PROVINCE, CHINA.

east, through mine No. 4 and the hamlet Sin-hung-chin. All the above sketches are only approximately to scale.

These illustrations show five mines, numbered consecutively from west to east. Up to the time of my visit there had been no deep mining, all workings being comparatively shallow open-cuts above drainage. Mine No. 1 proved to be on a small deposit, and had been worked out and abandoned. Mine No. 2 was still worked, but in a small way only, the ores being largely manganiferous brown ores. Mines Nos. 3, 4, and 5 were closely grouped together, apparently on a common lens of ore, and constituted the main workings in 1907. Explorations further to the east had discovered additional ore-deposits, but these had at that time not been developed. Read, as quoted above, says that ores have also been found along the northern contact of the granite and limestone.

We note here that the contact is only intermittently ore-bearing, the deposits occurring in a succession of lenses of varying size, with barren stretches between.

I examined briefly the ore-body revealed in mines Nos. 3, 4, and 5. This lens was from 1,500 to 2,000 ft. long and from 100 to 200 ft. thick, and had been excavated to a maximum depth of probably not more than 75 ft. below the original crop. The strike is approximately E–W., with the contact, and the dip from 65° to 85° N. Owing to the northward dip, the granite, in mining terminology, constitutes the hanging-wall, and the marble is the foot-wall. The granite wall, while fairly persistent in its E–W. strike in a general way, is irregular and somewhat angular in detail. The actual line of contact between the ore and the granite, where it could be observed in a fresh condition, was seen to be sharp. On the foot-wall side the transition-line from ore to marble was again sharp, and even more irregular in detail than on the hanging-wall. In this case the irregularities were curved rather than angular, however, and close to the wall many rounded inclusions of marble were seen in the ore.

The ores filling the space between the two walls were for the most part massive, though a certain amount of shearing parallel with the contact was observed. They include hematite and magnetite, the latter being present in a very considerable proportion, and being, in my opinion, primary. Further, there are smaller amounts of the hydrated oxides in the more weathered portions of the vein. The mass, as a whole, is high-grade, gangue-minerals appearing in rather subordinate amounts. These include quartz, some amphibole and greenish mica, occasional epidote, and more commonly copper carbonates, with rare sulphides of copper and iron. In Table I. I have brought together a number of analyses from various sources.

TABLE I.—*Analyses.*

	(1)	(2)	(3)			(4)	(5)
			a.	*b.*	*c.*		
Fe.........	58 to 68	60 to 62	62.0	48.0	65.0	59 to 65	65.5
SiO_2........	7 to 3	5 to 3	4.0	6.0	3.0	10 to 3.5	3.5
Al_2O_3.....	1 to 2	1 to 2	2.0				
Mn.......	0.2 to 0.4	0.2 to 0.4	0.3	7.0	0.13	....	0.10
P..........	0.04 to 0.25	0.05 to 0.25	0.05	0.03	0.12	0.04 to 0.10	0.04
S..........	0.05 to 0.10	0.05 to 0.12	0.02	0.01	0.06	0.10	0.05
Cu.........	0.05 to 0.25	0.05 to 0.25	....	....	....	0.20	0.09

(1) From *Iron Age*, (4) p. 1436. (2) Read, (1) p. 320. (3) Seltzer, (8) p. 1233—*a*, hematite; *b*, brown ore; *c*, magnetite. (4) Reported to me by officials at the Hanyang iron- and steel-works. (5) Reported to me by officials at the Japanese government iron- and steel-works.

The evidence at hand bearing upon the geological horizon at which the Ta-yeh deposits occur consists briefly of the following facts:

The limestone associated with the ore-deposits is closely overlain by red beds. These appear at Huang-shi-kiang, Fig. 1, and continue thence for some distance up the river. There are also conspicuous red cliffs several miles down the river. Furthermore, thin coal-beds are reported to exist within a few miles NE., W., and SE. of Ta-yeh.

Willis[6] gives the following geological column for the Yangtze gorges, about 200 miles westward:

Permo-Mesozoic: K'ui-choú series.
1,000 ft. sandstones, shales, and sandy shales with thin limestones and coal-seams.
150 ft. gray fossiliferous limestone.
400 ft. massive red shale.
Unconformity inferred.

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[6] *Research in China*, Willis, Blackwelder, and Sargent, vol. i., part i, p. 264. Carnegie Institution, 1907.

Upper Carboniferous: Wu-chan limestone.
3,000 ft. massive thick-bedded limestone of gray-brown or black color; locally flinty.
1,000 ft. shales and limestones, the latter locally rich in flint nodules.

Below this come 1,800 ft. of Sin-t'an shale (Middle Palæozoic), and about 4,000 ft. of Sinian (Cambro-Ordovician) Ki-sin-ling limestone, with some minor beds of shale and conglomerate, overlying the basal Algonkian (?) gneisses.

Elsewhere Willis cites Kingsmill with reference to localities near the Yangtze river as follows:[7] "... he [Kingsmill] distinguishes two horizons at which coal beds occur; one, the 'lower coal and iron shales,' 600 to 800 feet thick, being near the middle of his 'Tung-t'ing' series and dividing the great Carboniferous limestone in two; the other, the upper member of his 'Chung-shan' series, being higher up in the geologic column and in all probability of the same age as the early Jurassic (Rhetic) coals of the Middle Yang-tzi."

Comparing the above sections with the observed and reported conditions in the vicinity of Ta-yeh, it seems a reasonable inference that the Ta-yeh limestone is near the top of the Wu-chan (Upper Carboniferous); that the red beds at Huang-shi-kiang as well as down the river correspond to the "massive red shale" at the base of the Permo-Mesozoic; and that the adjacent thin-coal areas, which practically surround the granite ellipse, belong to the Permo-Mesozoic (early Jurassic) coals of Willis, corresponding to the "Chung-shan" coals of Kingsmill. This inference would argue a quaquaversal dip of the sediments about the granite.

In connection with this inquiry into the probable age of the associated limestone, I find of interest a description of certain Kiang-su province iron-ore deposits, written 50 years ago by Markham, then British consul at Chin-Kiang, a town on the Yangtze lying about 35 miles ENE. from Nanking. This description has been preserved by Kingsmill and is quoted by Willis.[8]

Markham describes three localities where iron-ore has been found. One of these, an inferior deposit within 3 miles of the gates of Nanking, is definitely referred to beds "at the foot of the Chung-shan." The other two, being respectively 5 and 14 miles distant from Chin-Kiang, appear to be probably below Kingsmill's "lower coal and iron-shales." While, therefore, the first is at about the same horizon as the Ta-yeh ore-bodies, the others may be from 1,000 to 1,500 ft. lower down in the geologic scale, though still in the Upper Carboniferous. Both these last are described in association with chert-bearing limestone, in one case there being present a large intrusive mass of porphyry.

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[7] Mineral Resources of China, *Economic Geology*, vol. iii., No. 1, p. 32 (Jan.–Feb., 1908).

[8] *Loc. cit.*, p. 35.

My brief inspection of the Ta-yeh deposits was quite insufficient to warrant any attempt on my part at an estimate of the probable available tonnage of ore, though this would appear to be very considerable. Mr. Read and others have presented descriptions and photographs of the generally primitive methods of open-cut hand-mining, and I find little in my notes to add to what has already been written in this direction. I, therefore, turn at once to the question of genesis.

In quoting from Read I have tried to include all his more significant references to the ore-deposits from a possible genetic standpoint. Whereas, the contact is recognized as being between an intrusive igneous rock and an intruded sediment, the genesis of the ores would appear to be referred rather to some cause other than the intrusive. There is no definite statement under this head, but two things are suggestive of such a trend of view, namely: the point is brought out that the deposits of Hu-pei (Ta-yeh), An-hwei, and Kiang-su provinces are probably at a common stratigraphical horizon; the magnetite is referred to as secondary. The first of these would naturally suggest that there is held to be some definite horizon in the limestone which, due to peculiar inherent or closely associated conditions, would carry ore-bodies to the exclusion of other horizons; in other words, that the ore is due to the presence of a definite horizon in the limestone, and not to the accidental association of this or any other horizon of limestone with an intrusive. The second point would suggest the opinion that the ore has been derived from meteoric, rather than deep-seated, sources.

There can be little doubt as to the nature of the contact. It is decidedly not a natural one. If faulted, then a fault with a throw of several thousand feet would be required to bring the Upper Carboniferous limestone into juxtaposition with the basement Archæan. So profound a fault would surely have left a corresponding stamp upon the adjacent strata. Such shearing as was observed was in the ore-body and, therefore, due to movement subsequent to its deposition. The granite gives no evidence of profound faulting, and the overturning of the limestone strata at the contact can be more readily accounted for otherwise. The contact is, in fact, undoubtedly one of intrusion, and in the granite we find a great laccolith which has invaded the limestone at some depth, and has since been uncovered by erosion. The elliptical area of the granite and the apparent quaquaversal dip of the sediments are in line with this supposition, while the structural conditions actually observed along the southern contact can well be explained by it. The limestone shows evidence of heat-metamorphism. The rarity of contact-minerals in the zone which came under my observation may be considered strange, but, as already pointed out, the intrusive and sedimentary members were separated at that point by 200 ft. of iron-ore. A study of the contact along the stretches barren of ore should be full of interest.

The source of the iron which has been deposited along this intrusive contact is our next consideration.

That the iron has been derived from the limestone, or from any particular horizon in the limestone, or that the ore-bodies represent the metamorphosed upturned edge of bedded veins of ore original in and conformably intercalated with the limestone, are suppositions which appear to me highly improbable. There is no appearance of such a possible source for the iron, or original existence of the vein, either in the character of the limestone beds, or in their position relative to the deposits. Read's suggestion that the deposits of the three provinces, Hu-pei, An-hwei, and Kiang-su, are at a common stratigraphical horizon would, if true, lend support to some such view. I have indicated, however, that the ores of the several provinces are more probably not at a common horizon.

A more plausible hypothesis, in my opinion, would be that the iron of Ta-yeh has been derived out of the once overlying but now eroded red beds, and has been transported in solution by descending meteoric waters along trunk-channels developed at the contact, till deposited as opportunity favored, either in pre-existing cavities or by metasomatic processes, or both. The rôle played by the intrusive rock in this hypothesis would be merely to have furnished at its more or less shattered contact a plane of weakness favorable to the passage of meteoric solutions. The magnetite found as a considerable part of the ore-bodies would then have to be accounted for, as Read suggests, by the supposed action of subsequent reducing solutions.

The position of the ore-deposits, however, is adverse to this theory. We find them underlying an intrusive contact. After intrusion and all its after-effects had ceased, the contact was probably not an open one; nor is there anything in the nature of an impervious foot-wall to guide the solutions or to retard and induce the precipitation of their burdens. Then, too, we should have to appeal to further processes to account for the nature of the ore and its gangue-minerals.

There is no evidence, such, perhaps, as a basic rim to the granite laccolith, to suggest that the ore-bodies are possibly due to ultrabasic magmatic segregations; nor does the character of the ore lend support to such a theory.

The natural hypothesis is that the ores are due to the pneumatolytic after-effects, or, as Stevens[9] has recently amended the term, the emanation residues, of the granite intrusion. The ores have been expelled out of the cooling granitic magma in gaseous, vaporous, or liquid solutions, and have been dropped along its contact, probably in part filling open spaces, caused either by the shrinkage of the granite mass in cooling or else by the partial dissolution of the limestone, or both, and in part replacing the limestone by metasomatic processes.

Ore-deposits of this type of genesis have now been recognized for a number of years and have been described from many localities. Leith has classified them under the designation of "Pegmatite Type."[10] Especially does his description of the iron-ore deposits of the Iron Springs District, Utah,[11] present many analogies with the Ta-yeh deposits.

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[9] Physical Data of Igneous Emanation, *Bulletin* No. 64, April, 1912, p. 435.

[10] Iron Ores of Canada, *Economic Geology*, vol. iii., No. 4, p. 277 (June-July, 1908).

[11] *Bulletin No.* 338, *U. S. Geological Survey* (1908).

The Ta-yeh iron-ore bodies are, in fact, a noteworthy example of this type of deposit, deserving of far more study than it was my opportunity to give to them. I feel that detailed study of this field should be rewarded by some valuable contributions to our knowledge of this interesting class of ore-bodies, while at the same time presenting some fascinating problems to the investigator.

SUBJECT TO REVISION.

[TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS.]

An Experience in the Use of Water-Power.

BY C. M. MYRICK, SAN FRANCISCO, CAL.

(Cleveland Meeting, October, 1912.)

THE following notes are submitted in the belief that they may interest some of the many owners of small water-power plants, so generally used in mining-work throughout the West.

A small and somewhat primitive quartz-mill was taken over by a leasing company, and a 10-stamp battery was installed. Originally, power had been furnished by a Pelton wheel; but, since the supply of water depended on the rate at which snow was melting on the mountains, a steam-plant had been added to help out during cold spells. This auxiliary plant was antiquated and expensive to operate, and, soon after starting up the remodeled mill, it broke down completely; so that it became necessary to get all the power possible out of the Pelton wheel, with its ever-varying water-supply.

The pipe-line, about a quarter of a mile long, was made up of assorted sizes, from 7 to 11 in. in diameter; and, since it was buried deep under snow, there were no data from which to calculate the theoretical size of the nozzle. The stream being at this time at a low stage, it was important to make the most efficient use of the available water. This was done by using a nozzle large enough to pass the whole supply—taking care, however, to limit the size so as not to lower the water-level in the tank at the head of the pipe-line.

As the weather became warmer, the nozzles were gradually enlarged to meet the increasing volume of water; but a point was finally reached where further enlargement only gave decreased power. Evidently the loss from friction had more than balanced the gain from the increased flow through the pipe. However, the area of nozzle that developed the maximum power from the pipe-line had been found. This was used; and the surplus water was allowed to run to waste at the head of the line.

All this experimenting necessitated the trial of nozzles of many different sizes; and, since these were not on hand, they were improvised as needed. Fig. 1 shows the quick and easy way in which this was done. A hole, 0.25 in. deep and of just the diameter of the outer rim of a cast-iron nozzle that had been found too large, was bored into a plank. Then, using a bit of the size desired for the new nozzle, the hole was continued through the plank. A tapering wooden plug, 8 or 10 in. long, was made to fit into this hole. The cast nozzle was forced into the shallow cut in the plank, the inner plug put in place, and the space between plug and nozzle filled with babbitt-metal. The wooden plug being removed left a nozzle of the desired diameter. When this particular size was no longer needed, the babbitt bushing was easily removed by heating.

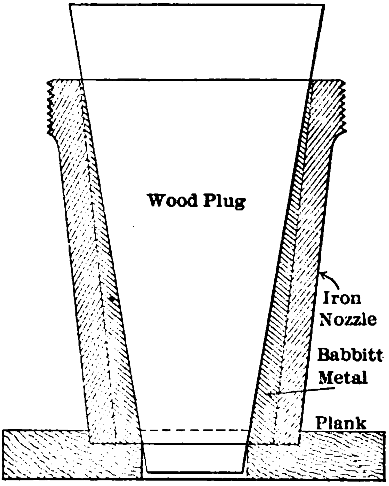

FIG. 1.—METHOD OF REDUCING THE SIZE OF A NOZZLE FOR A PIPE-LINE.

To determine the relative power furnished by the different nozzles, it was only necessary to count the number of drops of the stamps—a method which may seem somewhat crude, but which certainly showed the effective work being done.

In order to get a record of the pressure under the various conditions, the pipe was tapped close to the nozzle, and a steam-gauge, taken from the boiler, was attached. The note-book containing a memorandum of these tests was destroyed in the great fire at San Francisco; but the general facts are: that, with the valve closed, the static pressure in the pipe was 60 lb. per sq. in.; that, no matter how much water was available, it was harmful to use a nozzle larger than 2⅜ in.; and that even a change of $\frac{1}{16}$ in., either way, caused a perceptible slowing-down of the wheel.

Discharging through this 2⅜-in. nozzle, the pressure stood at 40 lb.; or, in general terms, water from this pipe-line was yielding its maximum power when one-third of the head was lost in overcoming pipe-resistances. Subsequent consultation of authorities showed that this agreed very well with the results obtained from the theoretical treatment of such problems.

Care was taken during the trial to lag up the pulleys, so that the circumferential speed of the water-wheel approximated to one-half the velocity of discharge from the nozzle, calculated from the effective head. It was noted, however, that a small variation was not important.

Incidentally, these experiments demonstrated that there was a very considerable waste of power if the cam-faces were not kept properly greased; and that this loss occurred long before the cams were dry enough to throw sparks. On the other hand, there was no necessity of excessive lubrication, incurring the risk of having grease thrown on the amalgamating-plates.

SUBJECT TO REVISION.

[TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS.]

The Sampling of Gold-Bullion.*

BY FREDERIC P. DEWEY, WASHINGTON, D. C.†

(Cleveland Meeting, October, 1912.)

AT the Seventh International Congress of Applied Chemistry I presented a paper,[1] The Assay and Valuation of Gold-Bullion, in which are briefly mentioned a few illustrations of different methods of sampling gold-bullion, particularly cyanide-bars. Since then I have carried on an extensive investigation upon the sampling of gold-bullion in relation to its effects upon the assay-results, in connection with the statement of accounts between the smaller assay-offices of the United States which purchase gold-bullion and the mints where the purchases are re-deposited.

Under the name of gold-bullion are included metals of widely different composition. We may have practically pure metal, 999.75 fine or even more in gold; alloys of gold and silver with only trifling amounts of base metal; alloys of gold and copper with trifling amounts of other metals; ternary alloys of gold, silver, and copper, or gold, silver, and lead; and most complex alloys of from five to eight or more constituents of ordinary occurrence, besides some of the rarer metals occasionally. In any one or all of these classes we may have one or more of the metalloids, which may add to our difficulties.

Of course, with metal 999.75 fine in gold there is only slight opportunity for different samples to differ in fineness, but on such fine metal we expect the assays to agree very closely, and the sampling must be done with the utmost care. When possible, dip- or granulation-samples of the molten metal should be taken; but when it is necessary to sample bars, the chief point to guard against is surface-dirt.

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* This paper was read at the Eighth International Congress of Applied Chemistry, New York, September, 1912, in advance of its appearance in the *Annual Report of the Director of the Mint for the Year* 1911–12, through the kind permission of the Director. It is here published by mutual agreement.

† Assayer, Mint Bureau, U. S. Treasury, Washington, D. C.

[1] *Trans.*, xl., 780 to 797 (1909), and in *Annual Report of the Director of the Mint for the Year* 1908–09, p. 25.

In the case of four melts of fine gold, which had been carefully sampled by both dips and chips, one of the chip-bars and an independent bar were chipped, and the chips assayed. Table I. summarizes the results.

TABLE I.—*Fine Gold Sampling.*

Original Report. Fineness.	Sample-Bar. 2d Sample. Fineness.	2d Bar. Fineness.
997.9	997.7	997.9
	998.	
997.5	997.4	997.4
	997.3	
997.4	997.4	997.3
	997.4	
997.2	996.9	996.9

In the case of four melts of metal of much higher grade, independent samples were taken, and assayed after the melts had been reported by the assayer. Table II. summarizes the results.

TABLE II.—*Fine Gold Sampling.*

Original Report. Fineness.	Resamples. Fineness.
999.8	999.7
	999.7
999.7	999.7
	999.7
999.7	999.7
	999.6
999.6	999.5
	999.5

Our standard gold (coin-gold) contains 900 parts of gold and 100 of copper. When made from pure metals it does not segregate. Four melts of standard gold were carefully sampled by both dips and chips and reported by the assayer. Subsequently one of the sample-bars and a second bar were chipped, and the chips were assayed. Table III. summarizes the results.

TABLE III.—*Standard Gold Sampling.*

Original Report. Fineness.	Sample-Bar. 2d Sample. Fineness.	2d Bar. Fineness.
900	899.8	899.8
	899.9	
899.9	899.8	899.8
899.8	899.7	899.6
899.7	899.7	899.5
	899.8	

In the cases already cited the sampling is a very simple matter, and the variations in the results shown may as well be due to the assaying itself as to the sampling. When, however, we come to consider miscellaneous bullion of more complex composition, the sampling assumes greater importance.

From a bar of strictly homogeneous metal, manifestly it would make no difference how the sample was taken, since any part of the metal would represent the whole. Bars that are practically homogeneous are not uncommon; and such bars would not have to be melted for the purpose of sampling, if we could be certain that they were homogeneous. Only two classes of alloys can form homogeneous bars—those that are solid solutions, and the eutectics; and unfortunately there are no characteristics by which the homogeneity of a bar of gold-bullion may be readily established. As a general proposition it is safe to assume that a brittle bar of gold-bullion will not be homogeneous. We may, of course, happen to get a bullion of eutectic composition, which would be both brittle and homogeneous, but eutectics undoubtedly form a very small proportion of the ordinary run of gold-bullion. On the other hand, it is far from safe to assume that a ductile bar is homogeneous.

The alloys of gold and silver, and gold and copper form continuous series of solid solutions, and they may carry small amounts of other metals without showing marked segregation. In several of the examples cited below, remarkable agreement in the assays of different samples of gold and silver alloys containing small amounts of impurities shows the practical absence of segregation. Alloys of gold and copper are quite similar in their heat-behavior to the gold-silver alloys, but I do not happen to have any satisfactory series of samples of these carrying small amounts of other metals.

When we come to the ternary alloys our knowledge of their behavior on heating and cooling is too scanty to permit valid generalizations, and this is even more emphatically true of the alloys of increasing complexity.

Of general miscellaneous gold-bullion, there is only one universally satisfactory method of taking a sample that shall truly represent the metal sampled. This consists in pouring a small portion of the well-mixed molten metal into water, so as to produce small globules or granulations of the metal. As this is generally done by dipping out a portion of the molten metal in a small cup, such a sample is frequently called a "dip," and this name is generally used throughout the present paper. These samples are, however, often called "granulations;" and they are sometimes made by pouring directly out of the crucible into the water, the operation of casting being interrupted for the purpose.

As already indicated, there are indeed various cases where other styles of sampling may be sufficiently satisfactory, and there are, moreover, many cases where it is desirable or even necessary to sample a bar of solid bullion without melting it. In such cases a chip may be cut off from the bar with an ordinary cold-chisel, or a chisel specially designed for the purpose. Power-driven punches with special tools may be used. Machines are also built which bite out a triangular piece of metal by means of a projection on a lever operated by a cam. Where much chip-sampling is done, especially on small bars, these machines save much time and labor. A third method of sampling consists in boring into the bar, generally with a power-drill, and using the drillings for the assay-sample.

There is but little choice in the location of a chip-sample. It must necessarily be taken from a corner or along an edge of the bar. In the systematic sampling of large bars, generally two chips are cut—one from the top and one from the bottom of the bar—and properly identified.

In taking drill-samples there is a wide choice in the location of the drill-holes and in the sampling of large bars more or less of a plan in placing the holes is often followed. It is a common practice in the Mint Service to drill half-way through a bar at diagonally opposite corners of the top, and unite the drillings for the top-sample. The remaining corners are drilled half-way through from the bottom, and the drillings are mixed for the bottom-sample. Occasionally, especially when sampling very base bars, the four drillings are kept separate; and sometimes holes are drilled near the center of the bar also.

Drill-samples are often more satisfactory than chip-samples, especially where large numbers of bars of fairly uniform size and composition are sampled in accordance with a well-designed plan. Drill-samples of brittle bars are, however, liable to be unsatisfactory, because the fine and coarse portions may differ considerably in composition.

In the purchase of gold-bullion by the Mint Service of the United States, the size of the deposit has an important bearing upon the question of sampling. A very large proportion of the deposits will weigh less than 100 oz. each. Manifestly, slight differences in the samples on such bars will be immaterial. When, however, the weight of a deposit reaches 300 oz. the samples become important, and with bars weighing from 700 to 1,200 oz. correct sampling is essential.

Limiting myself for the most part to these large bars of miscellaneous bullion, I propose to illustrate by specific cases, drawn from actual practice, some of the general principles underlying the sampling of gold-bullion. In considering these illustrations, the assay-results must not be too rigidly interpreted. Besides the variations in the results due to variations in the samples, there are three other variants that must be kept in mind constantly: the chapter of accidents; the personal equation of the assayer; and the effect of the composition of the metal upon the assaying as distinct from its effect upon the sampling. This last point will be specially dealt with by various illustrations. Making reasonable allowances for these variants, it is thought that the illustrations are sufficiently conclusive upon the main points involved.

To begin with alloys of gold and silver containing only small amounts of base metal, four bars, each one carrying less than 4 base, were chipped top and bottom, and each one of the eight samples was assayed in three laboratories. A fifth bar, carrying 10 base, was treated in the same way. Table IV. summarizes the results reported.

TABLE IV.—*Sampling Gold-Silver Alloys Low in Base by Chips.*

Fineness.	Assays.		Fineness.	Assays.		Fineness.	Assays.		Fineness.	Assays.		Fineness.	Assays.	
872.3	1	...	875.7	...	1	883.3	1	1	884.2	2	...	873.8	...	1
872.4	2	...	875.8	1	2	883.4	...	6	884.3	...	1	874.0	1	...
872.5	5	4	875.9	2	3	883.5	4	2	884.4	1	4	874.2	2	1
872.6	...	2	876.0	4	2	883.6	2	1	884.5	1	2	874.3	1	...
872.7	...	3	876.1	1	1	883.7	3		884.6	1	...	874.4	2	1
873.0	1	1	876.2	2	1	....	....		884.7	...	1	874.5	2	4
....	....		....	....		....	....		....	....		874.6	...	1
....	....		....	....		....	....		....	....		874.7	1	...
Total...	9	10		10	10		10	10		5	8		9	8
Silver...	124			121			114			111			116	

Each one of the above bars may be taken as a single product, but a bar obtained by melting 39 small bars of miscellaneous origin into a mass-melt exhibits a remarkable agreement of chip-samples. This mass weighed a little more than 1,300 oz. and was 143 fine in silver. It was chipped, top and bottom, at the assay-office, and twice at the mint of re-deposit. The assay-office samples were assayed in duplicate there, and the four mint samples were assayed in four laboratories; 44 assays were made on the six chip-samples. Table V. summarizes the results reported:

TABLE V.—*Sampling a Mass-Melt by Chips.*

1 assay showed	679.7 fine in gold.
4 assays showed	679.8 fine in gold.
2 assays showed	678.9 fine in gold.
2 assays showed	680.1 fine in gold.
5 assays showed	680.2 fine in gold.
8 assays showed	680.3 fine in gold.
3 assays showed	680.4 fine in gold.
4 assays showed	680.5 fine in gold.
7 assays showed	680.6 fine in gold.
5 assays showed	680.7 fine in gold.
2 assays showed	680.8 fine in gold.
1 assay showed	680.9 fine in gold.
44	

A bar carrying more than 90 base and weighing 606 oz. was chipped, top and bottom, and the samples were assayed at the purchasing office. At the receiving-mint it was drilled, top and bottom, and the drillings were assayed in four service laboratories. Table VI. summarizes these assays.

TABLE VI.—*Chip- and Drill-Samples Agreeing.*

Fineness.	Chip.	Drill.
741.6	...	1
741.7	...	1
741.8	...	1
741.9	...	2
742.0	...	2
742.1	2	3
742.2	2	1
742.3	...	4
742.4	...	5
Total	4	20
Silver	165	

A bar weighing more than 800 oz. presents another remarkable case of the presence of considerable base metal without marked segregation. This bar was drilled, top and bottom, and the drillings were united to form one sample. Two other sets of drillings, top and bottom, were taken from the same bar, and each sample kept separate. These five samples were assayed 69 times in five laboratories. Table VII. summarizes the results reported.

TABLE VII.—*Sampling Gold-Silver Alloy, Containing Considerable Base, by Drilling.*

6 assays showed	351.0 fine in gold.
10 assays showed	351.1 fine in gold.
13 assays showed	351.2 fine in gold.
14 assays showed	351.3 fine in gold.
15 assays showed	351.4 fine in gold.
6 assays showed	351.5 fine in gold.
2 assays showed	351.6 fine in gold.
3 assays showed	351.7 fine in gold.
69	

This bar was 604 fine in silver and 44.5 fine in base metals.

Two other bars containing less base metal, but also considerably less gold, yielded much less satisfactory results. These bars weighed nearly 1,200 oz. each. They were drilled, top and bottom, and the drillings united to form one sample. One bar was drilled twice again, top and bottom, and the other three times, top and bottom, each drilling being kept separate. These 12 samples were assayed in five laboratories and Table VIII. summarizes the results.

On the other hand, a ductile bar 778 fine in gold with only 9 base yielded unsatisfactory results for this grade of metal. This bar weighed more than 1,500 oz. and was drilled, top and bottom, the drillings being mixed for one sample. Being ductile, it was chipped twice, top and bottom, at the mint of re-deposit. These six samples were kept separate. The bar was remelted with a loss of 0.38 oz. and two dip-samples were taken. The cold bar was again drilled, top and bottom. These 11 samples were assayed in four service laboratories, and Table IX. summarizes the results reported.

TABLE VIII.—*Drill-Sampling Gold-Silver Alloy Low in Gold and Base.*

	1st Bar.						2d Bar.						
	1st Drill.	2d Drill.		3d Drill.			1st Drill.	2d Drill.		3d Drill.		4th Drill.	
Fineness.		T.	B.	T.	B.	Fineness.		T.	B.	T.	B.	T.	B.
290.2	...	...	...	2	...	290.6	1	...	...	...	...	...	...
290.6	1	1	...	2	...	290.9	1	...	...	...	...	...	...
290.7	...	1	...	2	...	291.1	1	1	...	...	...	...	...
290.8	...	1	...	2	...	291.5	1	...	...	...	...	...	...
290.9	...	2	...	2	...	291.9	1	2	...	...	...	...	...
291.0	...	...	...	3	...	292.0	1	2	...	...	...	1	...
291.1	1	1	...	1	...	292.1	2	...	...	...	...	...	...
291.2	...	2	...	...	...	292.2	...	2	...	...	2	2	...
291.3	...	2	...	1	...	292.3	1	1	...	...	1	3	1
291.4	2	2	1	...	2	292.4	...	2	1	...	4	2	3
291.5	4	...	1	...	...	292.5	3	2	2	...	2	3	3
291.6	2	1	...	...	2	292.6	4	...	4	...	1	1	2
291.7	2	...	...	...	...	292.7	...	1	4	...	1	1	3
291.8	...	...	...	...	1	292.8	...	1	1	...	1	1	...
291.9	1	...	1	...	1	292.9	...	...	...	...	...	...	2
292.0	...	...	...	...	1	293.0	...	...	2	1	1	...	...
292.1	...	...	1	...	...	293.1	...	...	...	4	1	1	...
292.2	2	...	2	...	...	293.2	...	1	...	4	...	...	...
292.3	...	...	2	...	1	293.3	...	...	...	2	...	...	...
292.4	...	1	3	...	...	293.6	...	...	...	1	...	...	...
292.5	...	...	1	...	...	293.9	...	...	...	1	...	...	...
292.7	...	...	1	...	1	...	...	...	...	...	...	...	...
292.8	...	...	1	...	2	...	...	...	...	...	...	...	...
292.9	...	...	...	...	1	...	...	...	...	...	...	...	...
293.0	...	...	1	...	1	...	...	...	...	...	...	...	...
293.1	...	...	1	...	1	...	...	...	...	...	...	...	...
Total...	15	14	15	15	14		16	15	14	13	14	15	14
Silver...		670								671			

TABLE IX.—*Sampling Ductile Gold-Silver Alloy, Low in Base.*

	1st Drill.	2d Drill.		1st Chip.		2d Chip.		Remelt.			
								Drill.		Dip.	
Fineness.		T.	B.	T.	B.	T.	B.	T.	B.	T.	B.
777.2	...	...	...	...	...	1	...	...	...	...	...
777.3	...	...	...	...	...	2	...	...	...	...	...
777.5	...	...	...	...	...	...	...	...	...	...	...
777.6	...	...	...	...	...	...	...	...	...	...	...
777.7	...	...	...	...	...	3	...	1	...	...	...
777.8	1	...	...	...	...	...	...	...	...	...	...
777.9	...	...	...	...	1	...	...	...	...	...	...
778.0	...	...	...	...	1			1	...	...	...
778.1	1	...	...	...	...	...	2	2	...	...	...
778.2	1	...	...	2	2	...	1	2	...	...	...
778.3	1	3	2	2	3	...	3	2	2	...	1
778.4	1	1	4	...	1	...	3	2	...	...	3
778.5	3	4	3	...	1	...	...	2	3	2	5
778.6	3	1	2	...	2	...	1	...	5	6	1
778.7	1	...	...	1	...	...	...	...	2	1	2
778.8	...	...	...	...	...	...	...	...	...	1	...
778.9	...	...	...	...	...	...	...	...	...	1	...
779.0	...	1	...	1	...	...	...	...	...	...	...
Total......	12	10	11	6	11	6	10	12	12	11	12
Silver......	212.5										

The following case exhibits what is probably the very best agreement between various samples, and between an assay-office and a mint, that can be attained under every-day working-conditions:

A deposit of approximately 3,000 oz. was melted in the assay-office, and two dip-samples were taken. It was cast into three bars, and two chip-samples were taken from each bar. At the mint of re-deposit, two chip-samples were cut from each bar. The mint samples were forwarded to the Bureau, and thence sent to the assay-office and afterwards returned to the mint to be assayed, neither institution being informed what the samples were.

In all, ten samples were taken from this deposit, and 47 assays were made at the two institutions. Table X. summarizes the assays reported.

TABLE X.—*Agreement of Assays on Various Samples at Two Institutions.*

1 assay showed	875.5 fine in gold.
1 assay showed	875.6 fine in gold.
4 assays showed	875.7 fine in gold.
4 assays showed	875.8 fine in gold.
14 assays showed	875.9 fine in gold.
12 assays showed	876.0 fine in gold.
6 assays showed	876.1 fine in gold.
4 assays showed	876.2 fine in gold.
1 assay showed	876.3 fine in gold.
47	

If we eliminate the results that were reported only once each, we have 44 assays, ranging from 875.7 to 876.2. This deposit might, therefore, be properly reported by either institution at either 875.75 or 876. It did not contain over 4 base.

On three bars the purchasing office took chip- and dip-samples, and on a fourth bar took dip- and drill-samples. The mint of re-deposit cut two chip-samples from each bar. These samples were assayed in various service laboratories, and Table XI. summarizes the results on the various styles of samples.

TABLE XI.—*Sampling in Various Ways with Satisfactory Agreement.*

Fineness.	Dip.	Chip.	Chip.	Fineness.	Dip.	Drill.	Chip.	Fineness.	Dip.	Chip.	Chip.	Fineness.	Dip.	Chip.	Chip.
861.4	...	1	2	895.1	1	...	2	858.5	1	...	1	883.2	...	...	1
861.5	...	...	1	895.2	...	...	3	858.7	...	1	...	883.3	...	2	...
861.7	1	1	...	895.3	...	1	4	858.9	...	...	1	883.4	1	1	1
861.8	...	4	1	895.4	1	...	2	859.0	1	3	1	883.5	2	1	6
861.9	3	1	4	895.5	1	2	1	859.1	1	1	2	883.6	...	...	1
862.0	3	3	6	895.6	1	1	2	859.2	3	3	7	883.7	...	1	4
862.1	3	1	3	895.7	2	2	1	859.3	1	1	2	883.8	5	4	...
862.2	1	1	...	895.8	3	3	...	859.4	3	...	1	883.9	3	3	3
862.3	2	1	...	895.9	3	4	...	859.5	5	1	...	884.0	3	2	1
862.4	1	...	...	896.0	1	...	1	859.6	...	3	3	884.1	...	...	1
......	...	...	...	896.1	1	...	...	859.7	1	...	...	884.2	...	...	1
......	...	...	...	896.2	...	1	...	......	...	...	...	......	...	...	...
Totals...	14	13	17		14	14	16		16	13	18		14	14	19
Silver...	134					98			137				118		

On three bars the purchasing office took dip- and drill-samples, and the mint of re-deposit took top and bottom drill-samples. These samples were assayed in various service laboratories, and the results are summarized in Table XII., showing a considerable agreement between the dip-samples and the drills taken at the separate offices. But a better agreement is desirable.

TABLE XII.—*Dip- and Drill-Samples Partly Satisfactory.*

Fineness.	Dip.	Drill.	Drill.	Fineness.	Dip.	Drill.	Drill.	Fineness.	Dip.	Drill.	Drill.
822.4	1	......	......	973.4	......	1	......	972.1	......	1	......
822.7	1	......	......	973.5	......	1	......	972.2	1	1	......
822.8	.....	......	2	973.6	......	......	1	972.3	3	......	3
822.9	1	1	1	973.8	......	1	1	972.4	4	1	......
823.0	3	1	2	974.0	......	......	2	972.5	......	......	1
823.1	3	2	1	974.1	......	1	......	972.6	3	1	1
823.2	2	4	......	974.2	2	......	2	972.7	1	......	2
823.3	.....	3	1	974.3	2	......	......	972.8	......	......	2
823.4	1	2	4	974.4	1	......	1	972.9	......	1	......
823.5	1	......	......	974.5	2	1	2	973.0	......	2	2
823.6	.....	......	2	974.6	1	......	......	973.1	......	3	......
823.7	.....	1	1	974.7	......	1	2	973.2	1	......	......
823.8	.....	......	4	974.8	3	2	4	973.3	1	1	1
823.9	.....	......	1	974.9	1	......	2	973.4	......	1	1
824.1	.....	......	1	975.0	......	4	1	973.5	......	......	2
.....	.....	......	......	975.1	1	1	......	973.6	......	......	1
.....	.....	......	......	975.2	......	1	2	973.8	......	1	1
......	.....	......	.....	975.3	1	......	1	974.3	......	1	......
.....	.....	......	......	......	......	......	......	975.7	......	......	2
Totals...	13	14	20		14	14	21		14	14	19
Silver...		162				4			Practically none.		

On four bars the purchasing office took dip- and drill-samples and the mint of re-deposit cut two chips from each bar. These samples were assayed in various laboratories in the service, and Table XIII. summarizes the results reported, showing the chip-samples to be unsatisfactory.

TABLE XIII.—*Chip-Samples Unsatisfactory.*

Fineness.	Dip.	Drill.	Chip.	Fineness.	Dip.	Drill.	Chip.	Fineness.	Dip	Drill.	Chip.	Fineness.	Dip.	Drill.	Chip.
821.8	...	...	1	892.3	...	...	1	898.0	...	...	1	820.7	1	...	
822.3	...	...	3	892.6	...	...	2	898.1	...	...	4	820.8	...	...	2
822.5	...	...	1	892.7	...	...	5	898.2	...	...	2	820.9	...	...	2
822.6	...	...	1	892.8	...	...	2	898.3	...	1	3	821.0	4	...	1
822.7	2	1	1	892.9	1	...	4	898.4	1	2	2	821.1	1	...	...
822.9	...	...	1	893.1	...	2	...	898.6	1	2	1	821.2	2	...	...
823.0	1	1	2	893.2	1	1	1	898.7	3	...	1	821.3	1	...	...
823.1	4	3	3	893.3	4	2	1	898.8	2	2	1	821.4	3	2	...
823.2	3	...	...	893.4	...	1	...	898.9	3	3	...	821.5	1	1	...
823.3	1	2	...	893.5	5	4	...	899.0	3	2	...	821.6	...	1	...
823.4	...	1	1	893.6	3	...	...	899.1	1	...	1	821.7	1	3	1
823.5	4	2	1	893.7	...	1	1	899.4	...	...	1	821.8	1	3	1
823.6	1	3	2	893.8	...	1	...	899.8	...	1	...	821.9	...	...	2
......	...	...	...	893.9	...	1	...	899.9	...	1	...	822.0	...	1	2
......	...	...	...	......	...	...	...	......	...	...	...	822.1	...	2	...
......	...	...	...	......	...	...	...	......	...	...	...	822.5	...	...	1
Totals..	16	13	17		14	13	17		14	14	17		15	13	12
Silver...		160				97				88				172	

On three bars the purchasing office took dip- and chip-samples and the receiving mint cut two chips. These samples were assayed in various laboratories in the service, and Table XIV. summarizes the results reported, showing that the mint chip-samples were entirely unsatisfactory.

A very complete set of samples from five bars, contained in one shipment made by a purchasing office to a mint for re-deposit, shows what may happen under every-day working-conditions in handling bullion of moderate grade. Everything considered, the first set of samples may be taken as fairly satisfactory, but the last set is totally unsatisfactory. At the office of purchase two dips and two drills were taken from each bar. At the mint of re-deposit two drills and two chips were taken. Each one of these eight samples on each bar was assayed in various service laboratories, and Table XV. summarizes the results reported; the assays on each set of two samples being grouped.

TABLE XIV.—*Second Chip-Samples Unsatisfactory.*

Fineness.	Dip.	Chip.	Chip.	Fineness.	Dip.	Chip.	Chip.	Fineness.	Dip.	Chip.	Chip.
879.8	...	...	1	890.3	...	...	1	867.6	...	...	1
880.0	...	...	1	890.5	...	...	2	867.8	...	...	1
880.1	...	...	2	890.6	...	...	2	868.1	...	...	2
880.6	...	...	1	890.8	...	...	1	868.2	...	...	1
881.3	...	...	1	890.9	...	...	4	868.3	...	...	1
881.4	...	...	1	891.0	...	...	1	868.5	...	...	4
881.5	...	...	2	891.1	...	...	1	868.6	...	...	4
881.6	...	...	1	891.2	...	...	1	868.7	...	...	1
881.7	...	...	2	891.3	...	2	1	868.8	...	1	...
881.9	...	1	...	891.4	2	...	...	869.0	...	1	1
882.0	...	1	1	891.5	...	2	...	869.3	1	...	...
882.1	...	...	1	891.6	4	1	...	869.5	...	1	...
882.2	...	2	...	891.7	5	1	...	869.6	1	5	...
882.3	2	3	...	891.8	2	4	...	869.7	1	1	...
882.4	3	1	...	891.9	...	5	...	869.8	2	1	...
882.5	2	4	...	892.0	4	1	...	869.9	1	1	...
882.6	2	...	1	892.2	1	3	...	870.0	1	2	...
882.7	4	...	...	892.3	...	1	...	870.1	3	1	...
882.8	...	3	...	......	...	...	...	870.2	1	2	...
882.9	2	...	...	......	...	...	...	870.3	4	2	...
883.0	3	1	...	......	...	...	...	870.4	1	...	...
883.1	1	1	...	......	...	...	...	870.5	2	1	...
883.2	...	1	...	......	...	...	...	870.6	1	...	...
......	...	...	...	......	...	...	...	870.7	1	...	...
......	...	...	...	......	...	...	...	870.9	1	...	...
......	...	...	...	......	...	...	...	871.0	1	...	...
......	...	...	...	......	...	...	...	871.1	...	2	...
Totals...	19	18	15		18	20	14		22	21	16
Silver...		90				98				95	

TABLE XV.—*Sampling Five Bars of Miscellaneous Bullion.*

Fineness.	Dip.	Drill.	Drill.	Chip.	Fineness.	Dip.	Drill.	Drill.	Chip.	Fineness.	Dip.	Drill.	Drill.	Chip.
630.3	...	1	1	...	755.0	...	...	1	1	544.2	...	...	...	1
630.4	...	1	...	1	755.1	...	...	1	...	544.3	...	...	...	1
630.5	...	...	...	2	755.2	...	...	...	2	544.6	...	...	...	1
630.6	3	...	..	...	755.3	...	...	1	2	544.7	...	...	...	1
630.7	1	1	2	3	755.4	...	...	3	2	544.9	...	...	...	1
630.8	...	...	2	1	755.5	...	1	1	...	545.0	...	...	...	2
630.9	2	1	2	1	755.6	...	...	1	4	545.1	...	...	1	1
631.0	2	2	2	2	755.7	...	...	...	1	545.2	...	...	...	1
631.1	3	...	...	...	755.9	...	1	1	...	545.3	....	...	...	2
631.2	2	2	...	1	756.0	...	...	1	...	545.4	1	...	...	1
631.3	1	2	1	...	756.1	...	1	1	1	545.5	..	...	1	...
631.4	1	2	...	...	756.2	...	2	3	1	545.6	...	...	1	1
631.5	1	2	1	1	756.3	...	1	...	...	545.7	...	...	1	...
631.6	...	...	2	...	756.4	...	1	...	...	545.8	...	1	1	1
631.7	...	1	1	...	756.5	1	...	1	...	545.9	2	...	1	...
......	...	...	...	...	756.6	2	...	1	...	546.0	2	...	...	...
.....	...	...	...	...	756.7	2	2	...	...	546.1	3	...	1	...
......	...	...	...	...	756.8	1	1	...	...	546.2	1	...	1	...
......	...	...	...	...	756.9	1	1	...	...	546.3	1	2	...	...
......	...	...	...	...	757.2	3	...	...	...	546.4	3	1	1	...
......	...	...	...	...	757.3	1	1	...	...	546.5	2	2	...	...
......	...	...	...	...	757.4	2	...	...	...	546.6	1	2	1	...
......	..	...	...	...	757.5	1	...	...	...	546.7	...	1	...	...
......	...	...	...	...	......	...	...	...	...	546.8	...	1	1	...
......	...	...	...	...	......	...	...	...	...	546.9	2	1	...	...
......	...	...	...	...	......	...	...	...	...	547.0	...	...	1	...
......	...	...	...	...	......	...	...	...	...	547.2	...	2	1	...
......	...	...	...	...	......	...	...	...	...	447.3	...	...	1	...
Totals...	16	15	14	12		14	12	16	14		18	13	14	14
Silver...			330					203					444	

Fineness.	Dip.	Drill.	Drill.	Chip.	Fineness.	Dip.	Drill.	Drill.	Chip.
549.4	...	...	...	1	568.5	...	...	...	1
549.7	...	...	...	1	569.6	...	...	...	1
550.5	...	...	...	1	573.9	...	...	...	1
551.0	...	...	...	1	574.6	...	...	...	1
551.4	...	...	...	1	575.4	...	...	...	1
551.7	...	...	...	2	575.7	...	...	...	1
551.8	...	...	...	1	576.4	...	...	...	1
551.9	1	...	1	...	576.5	...	...	...	1
552.1	1	...	...	2	577.0	...	...	...	2
552.2	...	...	1	...	577.2	...	...	...	1
552.3	...	...	...	1	577.3	...	...	1	...
552.5	...	...	1	1	577.4	...	...	...	1
552.6	1	...	3	...	577.6	...	...	1	...
552.7	3	...	2	1	577.8	...	...	1	...
552.8	2	1	1	...	577.9	...	...	1	...
552.9	4	4	3	...	578.0	...	...	2	...
553.0	2	...	...	...	578.1	...	...	1	...
553.1	4	5	1	...	578.3	1	...	...	...
553.2	3	3	...	...	578.4	...	1	...	...
553.3	...	3	...	...	578.5	...	1	1	...
553.4	...	1	1	...	578.6	...	3	2	...
553.5	1	...	...	...	578.7	3	2	2	1
553.6	...	...	2	...	578.8	3	1	...	...
......	...	...	...	...	578.9	4	2	1	...
......	...	...	...	...	579.0	4	1	1	...
......	...	...	...	...	579.1	...	3	...	...
......	...	...	...	...	579.2	1	1	...	1
......	...	...	...	...	579.3	2	...	...	...
......	...	...	...	...	579.4	1	1	...	...
......	...	...	...	...	579.5	1	...	...	...
......	...	...	...	...	579.6	...	1	...	...
......	...	...	...	...	579.7	...	1	...	...
Totals...	22	17	16	13		20	18	14	14
Silver....		410						335	

A small bar made by melting up scrap-material has a most interesting and instructive sample-history. The bar weighed a trifle more than 31 oz. and was shipped from the assay-office as 826 fine in gold. At the receiving mint it was drilled, top and bottom, and while the assays on these samples agreed fairly well, yet they averaged considerably less than 826. The bar was again drilled, top and bottom, and these samples yielded very erratic assays. The bar was then melted and two dip-samples were taken. The solid bar was again drilled, top and bottom. These eight samples were assayed in four service laboratories, and Table XVI. summarizes the results reported.

TABLE XVI.—*Sampling a Small Unsatisfactory Bar.*

					Remelt.			
	Drill.		Drill.		Drill.		Dip.	
Fineness.	T.	B.	T.	B.	T.	B.	1st.	2d.
823.0	......	......	......	1	......	......	......	......
823.5	......	......	......	1	......	......	......	......
823.7	......	......	......	1	......	......	......	......
823.9	......	......	......	1	......	......	......	.....
824.3	......	......	......	1	......	......	......	......
824.4	......	......	......	2	......	......	......	......
824.7	......	1	......	......	......	......	......	......
825.1	......	......	......	......	......	......	1	2
825.2	......	2	1	......	......	......	......	......
825.3	......	1	1	......	......	......	1	......
825.4	2	2	......	......	......	......	1	......
825.5	4	2	......	......	......	......	......	......
825.6	2	2	......	......	1	......	3	1
825.7	1	......	......	......	......	1	......	1
825.8	1	......	1	......	......	3	3	......
825.9	......	......	1	......	......	1	1	1
826.0	1	1	1	......	2	......	3	......
826.1	......	......	......	......	1	1	......	......
826.2	......	......	......	......	2	2	......	2
826.3	......	......	2	......	......	1	......	2
826.4	......	......	......	......	1	......	......	1
826.5	......	......	1	......	1	......	......	2
826.6	......	......	2	......	2	......	......	......
827.0	......	......	1	.....	......	......	......	......
Totals......	11	11	11	7	10	9	13	12
Silver......			143					

On three bars drills were taken from diagonally opposite corners on top of the bar and also near the center of the bar. The other two corners were drilled from the bottom of the bar and a drilling was made near the center of the bottom. All the top-drillings were mixed to make one sample, and all the bottom-drillings to make another. These samples were assayed in two laboratories, and Table XVII. summarizes the results reported, to which are added for comparison duplicate assays of two dip-samples of each bar made in one of the laboratories.

TABLE XVII.—*Sampling by Top and Bottom Drills.*

	1st Bar.			2d Bar.			3d Bar.	
Fineness.	Bottom.	Top.	Fineness.	Bottom.	Top.	Fineness.	Bottom.	Top.
883.7	2	......	884.8	3	......	888.7	1	......
883.9	4	......	884.9	3	......	888.8	1	......
884.0	1	......	885.0	4	......	888.9	1	......
884.1	6	......	885.1	3	......	889.0	2	......
884.2	3	......	885.2	3	......	889.1	3	......
884.3	1	......	885.3	1	......	889.2	2	......
884.4	1	......	885.4	5	......	889.3	1	......
884.5	1	......	885.5	2	......	889.4	2	......
884.7	......	1	885.6	1	......	889.5	2	1
885.1	......	5	885.7	1	......	889.6	1	......
885.2	......	2	886.2	......	4	889.7	1	......
885.3	......	1	886.3	......	3	889.8	3	......
885.4	......	3	886.5	......	6	890.0	......	2
885.5	......	2	886.6	......	4	890.1	......	3
885.6	......	4	886.7	......	1	890.2	......	2
885.7	......	2	886.8	......	1	890.3	......	1
......	......	......	887.0	......	1	890.6	......	2
......	......	......	......	......	......	891.1	......	1
......	......	......	......	......	......	891.3	......	1
......	......	......	......	......	......	891.4	......	1
......	......	......	......	......	......	891.6	......	1
......	......	......	......	......	......	891.7	......	1
......	......	......	......	......	......	891.8	......	1
......	......	......	......	......	......	891.9	......	1
......	......	......	......	......	......	892.2	......	2
	19	20		26	20		20	20

Dip-Samples.		Dip-Samples.		Dip-Samples.	
884.0	3	885.3	1	889.1	2
884.1	1	885.4	1	889.2	2
		885.5	1		
		885.7	1		

With the exception of the single assay at 889.5 on the third bar, all the top-assays are above the highest bottom-assay on each bar. All the top-assays are above the dip-assays in each one of these cases. Each one of the bars was less than 5 fine in silver.

As a general proposition, a drill-sample will often be better and more satisfactory than a chip-sample, simply because it represents a larger volume of the metal; but a drill-sample of a brittle bullion may be open to a serious objection, because the fine particles of the sample are generally liable to be quite different in composition from the coarse portions, so that, even if the drill-sample as a whole should be fairly representative of the bar, yet it would be necessary to have the 0.5 g. weighed out for the assay composed of fine and coarse material in just the same proportions as the whole sample. Manifestly this will seldom be the case.

Drill-samples of two closely-related bars were sifted on an 80-mesh screen, and yielded the following results:

	1st Bullion.	2d Bullion.
Coarse portion................	290.3 fine.	291.4 fine.
	290.4 fine.	391.5 fine.
Fine portion..................	287.8 fine.	290.1 fine.

Two other drills of these same bars were sifted. The portion between 20- and 40-mesh and that finer than 60-mesh were assayed, with the following results:

	1st Bullion.	2d Bullion.
20–40-mesh	290.4 fine.	291.9 fine.
	291.9 fine.	292.3 fine.
60-mesh	289.6 fine.	290.3 fine.
	290.3 fine.	290.8 fine.

Two other bullions treated like the last illustration showed:

	1st Bullion.	2d Bullion.
20–40-mesh...........................	351.7 fine.	573.2 fine.
	351.9 fine.	573.4 fine.
60-mesh..............................	350.2 fine.	566.6 fine.
	350.5 fine.	567.3 fine.

An entirely different bullion sifted differently showed:

40–60-mesh.........	585.3 fine.	100-mesh.........	584.6 fine.
	585.4 fine.		584.7 fine.

A very rich bullion sifted like the last showed:

40–60-mesh.........	889.2 fine.	100-mesh.........	887.8 fine.
	889.2 fine.		

Five samples of a bullion which had given discordant assays were subjected to sifting-tests, and the resulting samples were assayed. Four drill-samples were sifted on 100-mesh, and the coarse and fine portions yielded the following results:

A.		B.		C.		D.	
Coarse.	Fine.	Coarse.	Fine.	Coarse.	Fine.	Coarse.	Fine.
615	612.7	615.3	612.2	613.9	612.9	612.8	608.
617	614.4	615.9	612.3	615.1	613.1	616.6	609.4
		617.9	613.7		614.1		610.5

Slightly more than a gram of the small pieces of a dip-sample was crushed and assayed in duplicate. Coarser pieces were crushed and sifted on 80-mesh. The three samples yielded the following results:

Whole Dip-Sample.	Crushed Dip-Sample.	
	Coarse.	Fine.
614.8	614.8	614.8
615.	615.2	614.9

This bullion carried bismuth.

No class of bullion has given assayers the world over so much trouble as the so-called cyanide-bullion. Not all bullion produced by cyanide-mills is troublesome. Hundreds of bars produced by such mills have passed through our Mint Service without the slightest trouble. It is easy enough to produce a high-grade refined bar from the zinc-box precipitate; but when this precipitate is melted direct and put into bars without proper refining, there is liable to be no end of trouble with the assays. Unfortunately, too, in some instances the precipitate is not as thoroughly cleaned from zinc before drying as it might be. These dirty unrefined bars are the ones that are usually spoken of as "cyanide-bars" with so much disparagement.

More than 15 years ago Roberts-Austen[2] gave a startling illustration of the difficulties and uncertainties of assaying this class of bullion. A bar weighing 393 oz. was sampled and assayed in the usual manner, and paid for at £965. The gold from this bar was separated and refined by itself, and was found to be worth £1,028. This meant a loss of £63, or more than $300, to the former owner of the bar.

It is generally assumed that the zinc remaining in the bullion is the cause of the trouble, but no clear and systematic explanation of its action has yet been given. It is quite probable that its action is complex, and differs in different bullions according to the presence or absence of other metals. I have, for instance, published[3] 50 assays made in eight laboratories on a synthetic bullion approximately 590 fine in gold, 245 in silver, 130 in zinc, with a little copper and a very little lead, which ranged from 588.9 to 589.9 fine in gold. Clearly the zinc did not seriously interfere with the actual assaying in this case.

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[2] *Annual Report Deputy Master and Comptrollerof the Mint*, No. 27, p. 38 (1896).

[3] *Trans.*, xl., 794 (1909). *Annual Report of the Director of the Mint for the Year* 1908–09, p. 34.

I have also published[4] 207 assays made on three bars of bullion produced in the celebrated Mercur mill, showing widely-varying assays. By a qualitative analysis, this bullion was found to carry both cadmium and nickel, and four other bullions from cyanide-mills in Montana showed these metals. At present I am carrying on, as occasion permits, a series of test-assays on synthetic alloys of gold, zinc, and cadmium.

From our tests in sampling cyanide-bullion I am satisfied that much of the variation ordinarily shown by the assays of this class of bullion is due to differences in the samples, arising from the effect of the zinc upon the physical structure of the metal; but, aside from this, there are many cases where the composition of the metal directly affects the assay-work itself.

TABLE XVIII.—*Effects of Composition Upon the Assay-Work.*

Fineness.	1st Dip.	2d Dip.	Fineness.	1st Dip.	2d Dip.	Fineness.	1st Dip.	2d Dip.
784.9	......	1	816.6	......	1	821.9	1	.....
785.2	......	3	816.8	......	2	822.0	2	......
785.3	1	......	817.0	......	1	822.2	3	......
785.4	3	1	817.2	......	1	822.3	2	2
785.5	2	1	817.3	1	......	822.4	1	3
785.6	1	2	817.4	2	1	822.5	1	2
785.7	3	......	817.5	1	......	822.6	2	......
785.8	1	......	817.6	3	2	822.7	2	......
785.9	2	3	817.7	2	......	822.8	4	1
786.0	1	1	817.8	1	1	822.9	1	......
786.2	1	......	817.9	2	3	823.0	......	2
786.3	......	1	818.0	1	1	823.1	......	3
786.5	1	......	818.1	3	......	823.2	1	2
786.6	1	2	818.2	......	2	823.5	......	1
786.8	.....	1	818.3	1	1	823.6	......	2
787.0	1	3	818.4	1	......	823.7	......	1
......	......	......	818.5	......	2	......	......	......
......	......	......	818.6	......	2	......	......	......
......	......	......	818.7	......	1	......	......	......
......	......	......	818.8	1	1	......	......	......
......	......	......	819.0	1	1	......	......	......
......	......	......	819.1	......	1	......	......	......
......	......	......	819.2	2	......	......	......	......
......	......	......	819.3	......	1	......	......	......
Totals.........	18	19		22	25		20	19

These bars ranged from 5 to 10 fine in silver.

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[4] *Engineering and Mining Journal*, vol. xciii., No. 15, p. 733 (Apr. 13, 1912).

The illustrations of top- and bottom-drilling given in Table XVII. are from cyanide-bars. Five of these samples show that individual samples of cyanide-bullion may give fairly satisfactory results, but that two such samples from the same bar may leave the question of the actual content of gold in the bar in doubt.

TABLE XIX.—*Sampling Cyanide-Bullion.*

	1st Bar.			2d Bar.			3d Bar.	
Fineness.	Drills.	Dips.	Fineness.	Drills.	Dips.	Fineness.	Drills.	Dips.
826.9	1	......	816.9	1	......	831.5	1	......
827.2	2	......	817.2	1	......	832.1	......	1
827.3	1	......	817.3	3	......	832.3	......	3
827.4	1	......	817.5	1	......	832.4	......	1
827.6	1	......	818.3	......	1	832.6	......	1
827.7	3	......	818.4	1	......	832.8	1	2
827.8	1	......	818.5	......	1	832.9	1	1
828.3	......	1	818.6	1	1	833.0	......	4
828.4	2	......	818.7	......	1	833.1	1	3
828.5	4	......	818.8	......	1	833.3	1	2
828.6	1	1	818.9	1	2	833.4	1	3
828.7	1	4	819.0	1	1	833.5	3	6
828.8	2	1	819.1	2	1	833.6	1	1
829.0	1	3	819.2	......	1	833.7	1	1
829.1	2	2	819.3	......	1	833.8	2	1
829.2	3	3	819.4	......	1	833.9	......	1
829.3	1	5	819.5	1	3	834.0	1	2
829.4	1	2	819.6	2	3	834.1	1	2
829.5	1	5	819.7	......	2	834.4	1	1
829.6	1	3	819.8	......	2	834.7	1	......
829.7	......	1	819.9	......	4	835.0	1	......
829.8	.....	3	820.0	4	4	835.4	2	......
829.9	2	1	820.1	......	1	835.5	1	......
830.0	......	4	820.2	2	2	835.6	1	......
830.1	......	1	820.3	......	2	835.9	1	......
830.3	......	1	820.5	......	1	......	......	......
830.4	1	......	820.6	1	2	......	......	......
830.5	.....	1	820.7	2	1	......	......	......
830.8	1	......	820.8	2	......	......	......	......
......	......	......	820.9	2	......	......	......	......
......	......	.....	821.0	1	1	......	......	......
......	......	......	821.2	1	......	......	......	......
......	......	......	821.4	1	......	......	......	......
......	......	......	821.5	1	......	......	......	......
......	......	......	821.8	1	......	......	......	......
......	......	......	822.1	1	......	......	......	......
Totals......	34	42		34	40		23	36

All of these bars carried less than 5 of silver.

Again, three bars from the same mill were sampled by dips, twice each, and the six samples were assayed in various laboratories in the service, giving widely varying results, and showing the effect of the composition of the metal upon the actual assaying. Table XVIII. summarizes the results reported.

Three bars deposited at one time by a cyanide-mill furnish an excellent illustration of the general conditions in handling this class of bullion. Two drills and two dips were taken from each bar, and the 12 samples were sent to various laboratories in the service for assay. Table XIX. summarizes the results reported, together with the original assays at the office of deposit, the assays on each kind of sample being grouped.

In one office receiving a great deal of high-grade bullion it is the general experience that the chip-samples will run somewhat below the dip-samples on this rich material. Illustrations already given show that no fixed relation is generally exhibited between the dip- and drill-samples as to the tenor of gold. Where many bars of practically uniform composition are drilled in accordance with a well-designed plan, a more or less fixed relationship between the drill- and dip-assays might appear. Such a plan was developed at one time for certain deposits at one of our mints.

Table XX. shows that with cyanide-bars a drill-sample may give much higher assays than a dip-sample. On the first bar, 13 assays of the drill-samples are above the highest assay on the dip-samples; and on the second bar, 22 drill-assays exceeded the highest dip-assays. Two cyanide-bars, each weighing about 850 oz. and practically free from silver, were drilled, top and bottom, and these four samples, together with four corresponding dips, were sent to various service laboratories for assay. Table XX. summarizes the results reported, together with the original mint-assays, the assays on each kind of sample being grouped.

TABLE XX.—*Sampling Cyanide-Bullion by Drills and Dips.*

1st Bar.			2d Bar.		
Fineness.	Drills.	Dips.	Fineness.	Drills.	Dips.
894.7	1	......	905.6	1	......
894.8	2	......	906.1	2	......
895.0	1	......	906.2	2	......
895.2	1	......	906.6	......	1
895.3	......	1	906.8	1	1
895.5	1	......	906.9	1	......
895.7	......	1	907.0	1	......
895.8	2	2	907.1	1	1
895.9	1	......	907.2	......	2
896.0	1	......	907.3	......	2
896.1	1	1	907.4	......	2
896.2	2	1	907.5	1	2
896.3	3	3	907.6	......	2
896.4	2	1	907.7	2	4
896.5	2	3	907.8	1	2
896.6	3	1	907.9	2	5
896.7	2	3	908.0	2	......
896.8	1	2	908.1	1	......
896.9	1	2	908.3	1	1
897.0	1	......	908.4	2	......
897.1	......	1	908.5	2	......
897.2	......	1	908.7	1	1
897.5	1	......	908.9	1	......
898.0	1	......	909.0	1	......
898.1	2	......	909.4	1	......
898.6	1	......	909.5	3	......
898.7	1	......	909.6	2	......
899.2	1	......	909.7	1	......
899.4	1	......	909.8	1	......
899.6	2	......	909.9	1	......
899.8	3	......	910.3	2	......
......	......	......	910.4	1	......
......	......	......	910.5	1	......
......	......	......	910.7	1	......
......	......	......	910.8	1	......
......	......	......	911.0	1	......
......	......	......	911.4	1	......
......	......	......	911.8	1	......
......	......	......	911.9	1	......
......	......	......	912.0	1	......
	41	23		46	26

Undoubtedly much of the trouble with these cyanide-bars would be avoided by a better cleaning of the zinc-box precipitate. It is, however, easy enough to refine the impure bullion by strongly-oxidizing fusion in the crucible; but this is expensive in labor and reagents, and there is some loss of gold. I have previously published[5] an illustration of this.

A cyanide-bar weighing 643.30 oz. was melted, with a loss of 6.07 oz., and 10 assays on various kinds of samples were made, with the following results:

1 assay showed 844.6 fine in gold.
1 assay showed 846.3 fine in gold.
1 assay showed 846.6 fine in gold.
2 assays showed 847.0 fine in gold.
1 assay showed 847.2 fine in gold.
2 assays showed 847.6 fine in gold.
1 assay showed 847.8 fine in gold.
1 assay showed 848.0 fine in gold.
—
10

The bar was melted seven times, when it weighed 502.01 ounces, showing a total loss of 141.29 and an estimated loss of 3.75 oz. of gold. Eight assays were made on various samples of the final metal, with the following results:

1 assay showed 933.2 fine in gold.
2 assays showed 933.3 fine in gold.
2 assays showed 933.4 fine in gold.
1 assay showed 933.5 fine in gold.
2 assays showed 933.7 fine in gold.
—
8

The final bar was 21 fine in silver.

A great deal of old scrap material of the most heterogeneous character is purchased at some of our offices, and many of these deposits are small. On these small bars the question of sampling and assaying is not so important; but before the purchasing offices ship these small deposits to a mint they are united into mass-melts, when difficulties may appear in the sampling, and become important. Nine small bars were united to make a mass-melt of 338 oz. Dip- and drill-samples were taken at the purchasing office, and the bar was drilled, top and bottom, at the receiving mint. These samples were assayed in various service laboratories, and Table XXI. summarizes the results reported.

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[5] *Trans.*, xl., 789 (1909). *Annual Report of the Director of the Mint for the Year* 1908–09, p, 31.

TABLE XXI.—*Sampling Mass-Melt.*

Fineness.	Dip.	Drill.	Drill.
500.7	......	......	1
500.8	......	......	1
501.1	......	......	1
501.7	......	......	1
501.8	......	......	2
501.9	1	......	......
502.0	......	2	......
502.1	......	1	......
502.4	4	......	2
502.5	......	......	2
502.6	......	......	2
502.8	1	......	1
502.9	1	1	1
503.0	1	1	1
503.1	......	2	1
503.2	......	1	......
503.4	......	2	1
503.5	1	......	......
503.6	......	2	......
503.7	1	......	......
503.8	......	......	1
503.9	2	1	......
504.0	......	......	1
504.2	......	......	2
504.7	1	1	......
Totals........................	13	14	21
Silver........................		225	

What is probably the very worst case of disagreeing samples and assays ever investigated by the Mint Bureau was a bar weighing 774.39 oz., carrying nearly 400 base, largely copper, which was shipped to a mint as being 568 fine in gold. Three sets of drills, top and bottom, were taken, and, these proving very unsatisfactory, the bar was remelted, with a loss of 4.6 oz. Two dip-samples were taken, and the bar was again drilled, top and bottom. These ten samples were assayed in four laboratories in the service and 113 assays were made. Table XXII. summarizes these assays.

TABLE XXII.—*Sampling Bar 400 Base, Largely Copper.*

	Original Bar.							Remelt.			
	1st Drill.		2d Drill.		3d Drill.			Drill.		Dip.	
Fineness.	T.	B.	T.	B.	T.	B.	Fineness.	T.	B.	A.	B.
564.7	......	......	......	......	......	1	571.2	......	2	......	......
565.3	......	......	......	......	......	1	571.3	......	1	......	......
565.6	......	......	......	......	1	......	571.7	......	1	......	......
566.2	......	......	......	......	2	......	571.8	1	......	......	......
566.4	......	......	......	......	1	......	571.9	......	......	......	1
567.0	......	......	......	......	1	......	572.0	1	1	1	......
567.1	......	1	......	......	1	......	572.6	......	2	1	1
567.4	......	......	......	......	......	2	572.8	1	......	1	......
567.5	......	1	......	......	......	......	573.0	......	1	......	......
567.6	......	......	......	......	1	1	573.2	1	2	......	......
567.8	1	......	......	......	......	......	573.5	1	......	......	1
568.0	......	......	......	......	......	1	573.6	......	......	1	......
568.3	......	1	......	......	......	......	573.7	......	1	......	......
568.4	......	1	......	......	......	......	573.8	......	1	1	......
568.5	......	1	......	......	......	......	573.9	1	......	2	1
568.6	......	1	......	......	......	......	574.0	2	......	2	1
568.7	1	......	......	1	......	......	574.1	1	......	1	1
568.8	1	......	......	1	......	......	574.2	1	......	......	......
569.0	......	1	......	......	......	......	574.3	1	......	1	1
569.2	......	2	......	......	......	......	574.4	1	......	......	1
569.4	3	1	......	......	......	......	574.6	......	......	1	1
569.5	......	1	1	......	......	......	574.8	......	......	1	
569.6	......	......	......	1	......	......	575.0	......	......	1	2
569.7	1	1	......	1	......	......	575.5	......	.... .	......	1
569.8	......	......	1	......	......	......	575.7	......	......	......	1
569.9	......	......	1	......	......	......	576.0	......	......	......	1
570.0	......	......	......	2	......	......	....	......	......	......	......
570.1	1		......	......	......	......	....	......	......	......	......
570.2	1	......	2	......	......	......	....	......	......	......	......
570.3	......	......	......	1	......	......	....	......	......	......	......
570.4	......	......	1	......	......	......	....	......	......	......	......
570.5	1	......	......	1	......	......	....	......	......	......	......
570.6	2	......	2	......	......	......	....	......	......	......	......
570.7	......	......	2	1	......	......	....	......	......	......	......
570.8	......	......	......	1	......	......	....	......	......	......	......
570.9	......	......	1	......	......	......	....	......	......	......	......
571.0	......	......	......	1	......	......	....	......	......	......	......
571.3	......	......	......	1	......	......	....	......	......	......	......
Totals...	12	12	11	12	7	6		12	12	14	14

The presence of so much copper in this metal undoubtedly affected the actual assaying as well as the sampling of the bar.

In describing the taking of dip-samples I have spoken of the molten metal being "well mixed." This condition is absolutely essential to proper sampling. It is quite possible that in some of the cases already mentioned the molten metal had not been thoroughly mixed when the samples were taken. In the following case the molten metal certainly was not well mixed.

Two dip-samples were assayed in duplicate by me, showing a decided difference between the two samples. On a re-assay in duplicate, the same difference appeared. Each sample was also assayed once in two other laboratories. Table XXIII. summarizes the results:

TABLE XXIII.—*Differing Dip-Samples.*

	1st Sample.	2d Sample.
Bureau	615.6	612.8
	616.0	612.8
Re-assay	615.5	611.5
	616.3	611.1
2d laboratory	615.3	612.2
3d laboratory	615.5	611.9

Many melters, samplers, and assayers have an idea that unsatisfactory chip- and drill-samples can be corrected by simply melting the bar and taking fresh samples. There are, of course, cases where there was carelessness in the original melting and sampling, which can be cured by careful remelting and resampling, but in a vast majority of cases simple melting does little or no good. If there is a real difference of importance between two chips or two drillings or between chips and drillings of the same bar, it is due to the fact that the metal segregates on solidifying. This is a function of the chemical composition of the metal; and, under practical conditions, a remelting which does not materially change the composition can accomplish very little in reducing the segregation. If the chemical composition of the metal is such that it interferes with the actual assay-work, manifestly a remelting which does not change the composition of the bar will accomplish no good.

It therefore follows that when the assay-reports on an ordinary bar of miscellaneous bullion differ so widely as to be unsatisfactory, it should be melted and refined by strong oxidation. This will, of course, reduce the weight of the bar, but the loss in weight is nearly all base metal in most cases. The actual loss of gold will generally be small, except where a large amount, 50 oz. or more, of base must be removed, as in the case of many cyanide-bars, when the loss may become considerable. In the case already cited, the removal of 141.29 oz. from a bar weighing originally 643.30 oz. caused a loss of about 3.75 oz. of gold.

A bar carrying about 35 silver and weighing 494.26 oz. yielded varying assays on the gold, and was remelted with a loss of 0.63 oz. Again the assays were unsatisfactory, and it was remelted with a further loss of 1.09 oz. but without improvement in the assays. It was finally melted with a further loss of 5.20 oz. and still gave unsatisfactory assays. Two dip-samples were taken at each melting, and the eight samples were assayed in various service laboratories. Table XXIV. summarizes the results reported.

TABLE XXIV.—*Dip-Samples on Four Meltings.*

1st Melt.		2d Melt.		3d Melt.		4th Melt.	
Fineness.	Assays.	Fineness.	Assays.	Fineness.	Assays.	Fineness.	Assays.
806.0	1	809.9	1	812.8	1	818.3	1
806.1	1	810.1	2	813.0	1	818.4	1
806.3	2	810.6	3	813.2	2	818.8	1
806.4	2	810.7	1	813.3	1	819.1	1
806.5	1	810.8	2	813.6	2	819.2	1
806.7	1	810.9	1	813.7	1	819.3	1
806.8	2	811.0	2	813.8	1	819.5	1
806.9	3	811.1	1	813.9	1	819.9	1
807.0	2	811.3	1	814.2	1	820.2	2
807.1	1	811.6	2	814.6	2	820.5	3
807.2	3	811.8	2	814.8	2	820.6	1
807.4	2	812.2	1	814.9	1	820.8	1
807.5	2	812.3	1	815.0	3	821.0	1
807.6	3	......	......	815.3	1	821.3	1
807.7	2	......	......	......	......	821.4	2
807.8	3	......	......	......	......	......	......
807.9	1	......	......	......	......	......	......
808.0	1	......	......	......	......	......	......
808.1	2	......	......	......	......	......	......
808.2	3	......	......	......	......	......	......
808.3	1	......	......	......	......	......	......
808.5	2	......	......	......	......	......	......
Totals..	41		20		20		19

Another bar carrying about 35 silver and weighing 666.91 oz. was drilled at the ends and in the center, and the drills yielded the following results:

End Drills.		Center Drills.	
Fineness.	Assays.	Fineness.	Assays.
664.1	1	695.4	1
664.5	1	697.4	1
665.5	2	703.0	2

This bar was remelted and two dip-samples were taken, while the cold bar was again drilled at the ends and in the center. After melting, however, even with the loss of 33.29 oz., the metal remained unsatisfactory, as shown by the following assays:

Dips.		Drills.			
		Ends.		Center.	
Fineness.	Assays.	Fineness.	Assays.	Fineness.	Assays.
705.5	1	699.5	2	706.7	1
706.6	1	699.9	1	707.9	1
706.8	1	701.3	1	708.9	1
707.8	1	701.6	1	725.0	1
707.9	1	701.8	1	726.3	1
708.3	1			727.0	1
708.5	1				
708.7	1				

From these gold-assays the silver cannot be given with any exactness, but it was probably about 40.

CONCLUSIONS.

The foregoing data are summarized in the following conclusions:

In sampling deposits of miscellaneous gold-bullion weighing more than 300 oz.:

There are various cases where either a chip- or a drill-sample may be satisfactory.

There are various cases where a drill-sample is better than a chip-sample.

Where the assayer is acquainted with the metal, he may accept a chip- or a drill-sample.

On an unknown bullion it is unsafe to accept any sample except a properly prepared dip-sample.

In many cases, particularly of cyanide-bullion, the composition of the metal interferes with the actual assaying and the bullion must be refined before one can expect to determine the gold accurately.

SUBJECT TO REVISION.

[TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS.]

Centrifugal Machines for Ore-Grading and Ore-Concentrating.

BY GODFREY T. VIVIAN, CAMBORNE, CORNWALL, ENGLAND.

(Cleveland Meeting, October, 1912).

VERY often important discoveries are made in one industry that may be used to advantage in another, but, owing to the rarity that men step out of one industry into another, these discoveries remain unknown outside of the trade or profession in which they are made. The present paper is written solely with the object of bringing to the notice of mining and metallurgical engineers a centrifugal ore-concentrator which eventually may have a great application.

Ore-Grading.

Fig. 1 represents the action which takes place when solids of different dimensions are stirred up in a box and allowed to settle as minerals of the same size but of different specific gravity. This principle is applied practically in grading china-clay and in enriching tin-slimes. In the latter case the slimes are stirred around in a sieve with water and allowed to settle while at the same time the side of the sieve is struck with a hammer. The colloidal slimes are then poured off and various slices taken off horizontally. The great drawback of this process is that the particles are deposited horizontally and all the colloids are lost.

Fig. 2 represents the action in which the particles are deposited side by side. Here the drawback is the great length of the box needed in order to prevent excessive loss. The alternative to lengthening the box is to increase the force of gravity. Centrifugal force may be regarded practically as the force of gravity augmented.

Fig. 3 illustrates the Gee centrifugal machine for grading ores. The drum, *A*, fitted with a box, *B*, is mounted on a shaft or spindle, *C*, the whole being suspended from a ball-bearing of special design at *D*, supported between girders, *E*. Rotation is imparted by the pulley, *F;* a band-brake is fitted at *G*. The upper end of the drum is closed by a cap, *H*, which makes a water-tight joint with the drum at *I*, when clamped by the lock-ring, *7*. (This forms a bayonet joint.) The cap, *H*, has a hole in the middle and is held central on the spindle by means of the casting, *K*, which is a sliding-fit on the spindle, and is connected with the cap by the upper ends of six rods, *L*. At the bottom of the drum, *U*, is fitted a weir-plate or diaphragm. Depending from the cap into the drum is a cage, seen best in the section, consisting of six vertical square rods, *LL*, to which are attached the radial blades or vanes, *N*. These blades extend the whole length of the drum, being connected to the cap at the top and to a circular plate, *O*, at their lower ends.

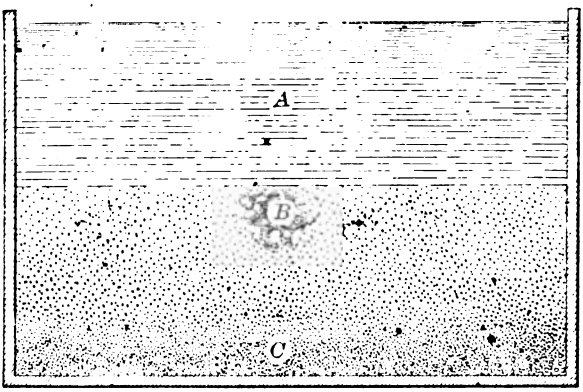

FIG. 1.—SETTLEMENT OF PARTICLES OF SAME SIZE BUT DIFFERENT SPECIFIC GRAVITY IN HORIZONTAL LAYERS.

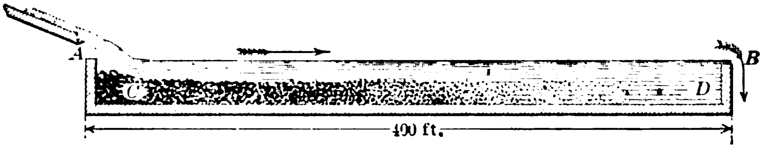

FIG. 2.—SETTLEMENT OF PARTICLES SIDE BY SIDE.

Fig. 4 shows the container clearly. This container slides easily on the drum, which it divides into six longitudinal compartments, each of which is provided with a curved plate. The container is in effect a movable lining to the drum, on which the recovered solids are received.

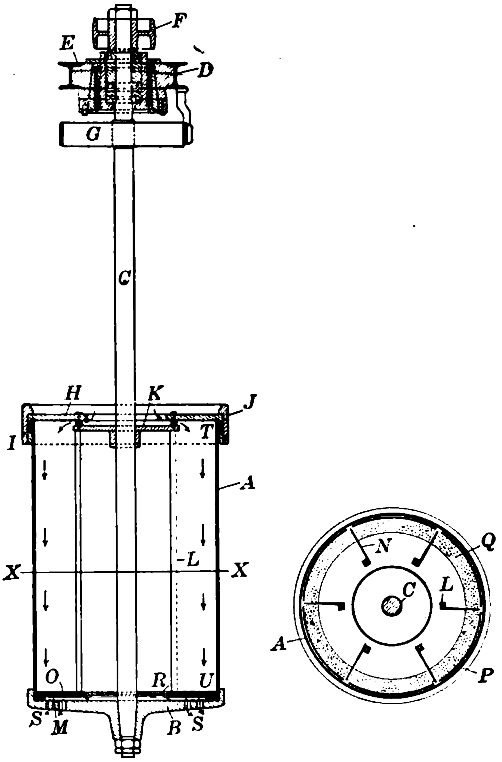

FIG. 3.—THE GEE CENTRIFUGAL ORE-GRADING MACHINE.

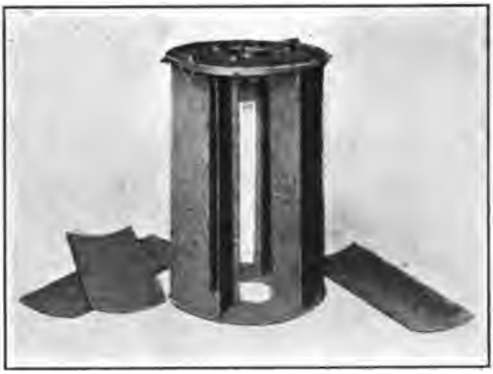

FIG. 4.—THE CONTAINER, OR DRUM-LINING.

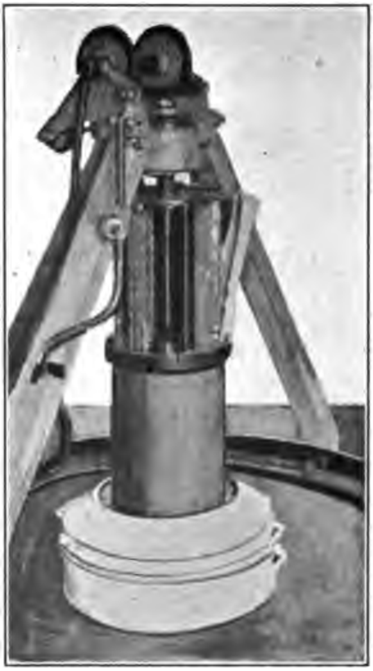

FIG. 5.—VIEW SHOWING REMOVAL OF SLATS FROM THE CONTAINER.

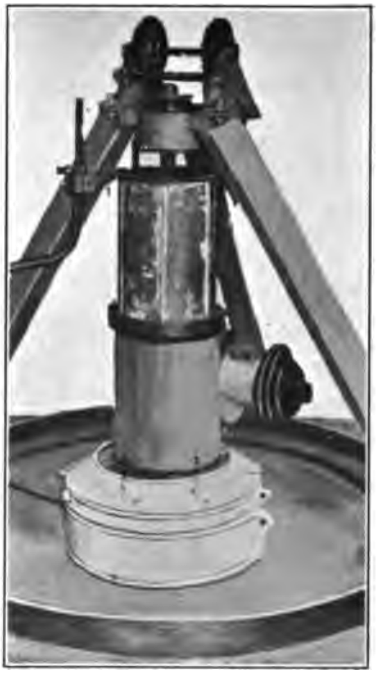

FIG. 6.—VIEW SHOWING CONTAINER IN RAISED POSITION.

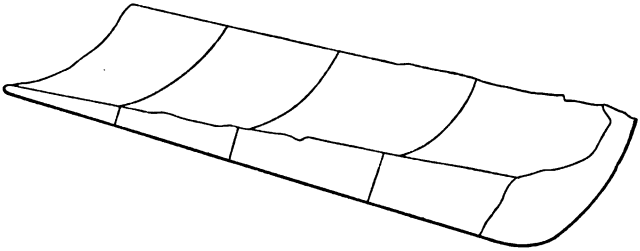

FIG. 7.—CAKE DIVIDED FOR GRADING.

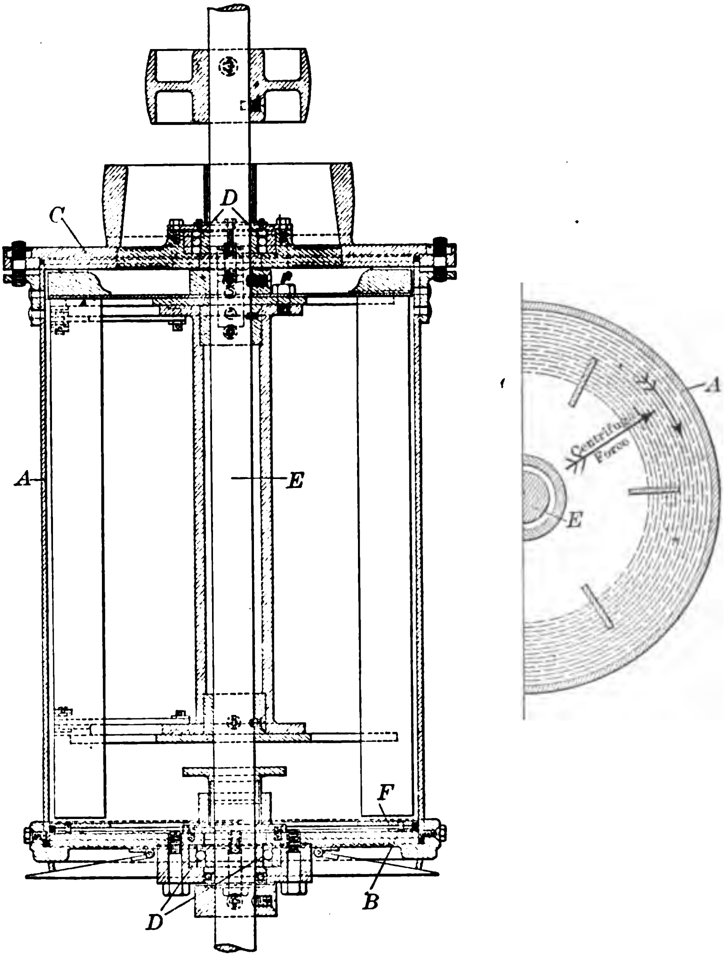

FIG. 8.—THE GEE CONCENTRATING-MACHINE.

Fig. 5 shows the manner in which the slats are removed.

Fig. 6 shows the container in a raised position, either before or after removal of cake.

Fig. 7 shows the cake divided by imaginary lines for grading.

Apart from grading china-clays, pigment-material, graphite, etc., the question which every mining and metallurgical engineer will ask is, How will the machine behave with gold-slimes? The cake is evidently drier than the pulp falling from a vacuum-filter. So far as I am aware, the experiment has never been tried practically.

Ore-Concentrating.

The concentrating-machine is illustrated by Fig. 8.

A is mounted on the base, *B*, and closed at the top by the cap, *C*. Ball-bearings, *DD*, at the top and the bottom of the drum are placed so that the drum may rotate on, but independently of, the shaft or spindle, *E*. As this machine is run at very much lower speed than the grading-machine, the bearings are placed at both ends. The weir-plate at the bottom, inside the drum at *F*, serves the same purpose as in the grading-machine, namely, to provide an inner wall of water of the desired thickness within the drum. Attached to the shaft, *E*, are a number of vanes or blades, extending the whole length of the drum, and so arranged that the distance they dip into the wall of water may be varied, so that the grip of these blades on the water may be adjusted.

The principle of the operation is, that the drum is caused to rotate at a given speed and the wall of water is caused by the vanes to rotate within the drum at a greater speed, but in the same direction. These speeds and their ratio to one another may be adjusted within very wide limits.

The result is that the particles of greater specific gravity, for instance, tin oxide with a gravity of 6.7, are, by reason of their greater inertia, caused to deposit on the drum and remain there, while the particles of lesser gravity—let us say quartz, 2.68 (?)—are discharged. It will be seen from this that the force of gravity as generally operating in concentrating-machines is practically augmented 100 times. There are five different factors in adjusting the apparatus: (1) The speed of the drum. (2) The speed of the water. (3) The grip of the vanes (forced by moving them towards or away from the drum). (4) The thickness of the wall of water. (5) The rate of flow through the drum, which varies the time during which the material is being acted on.

It will be at once inferred that this is a machine to be used on very fine material. Laboratory-experiments have been made with tin-slimes, which in practice are absolute slimes—giving a total extraction of from 95 to 97 per cent. While the results of these experiments are better than those that can be obtained in practice, these centrifugal machines may offer a commercial solution to the present slimes-problems and are such as warrant a further investigation.

I beg to thank William Gee, Steinhoff, Sons & Muir, Ltd., of London, for their courtesy in supplying information.

[TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS.]

Our National Resources and Our Federal Government.

BY R. W. RAYMOND, NEW YORK, N. Y.

(Cleveland Meeting, October, 1912.)

UNDER the names of Conservation, Social Justice, the New Nationalism, and Progressive Democracy, many earnest reformers are calling for a new system of Federal government to replace the one which they ascribe to our fathers, and declare to have been outgrown. They are proposing the lease by the United States of certain natural resources on the public domain, for the profit of the U. S. Treasury, instead of the sale of such resources to private citizens or associations of citizens. In the present paper, I shall try to show that this notion is neither new nor good.

It is clear that the United States is owner, as well as sovereign, of public lands acquired through conquest or purchase from other nations, as it was owner of the public lands expressly ceded to it by the original thirteen States. Its policy with regard to such lands involves, therefore, two entirely distinct questions: What shall it do as owner? and, What shall it do as sovereign? The latter question, it must be confessed, has played but a small part in our history. All the prerogatives of sovereignty which were not explicitly conferred upon the Federal government by the original States were retained by them. The ownership or control of precious or other metals or minerals in the earth, for instance, was thus reserved to the several sovereign States. New York, South Carolina, and other States have passed laws on this subject, without regard to Federal authority. As new States have been successively admitted to the Union, the tacitly accepted theory has been, that they thereby acquired all the reserved rights of the original States. These newer States largely outnumber the first thirteen; and it is inconceivable that they would submit to any deprivation of sovereignty based on historical grounds. One of them—Texas—came into the Union by its own voluntary act, as an organized commonwealth, already owning the whole of its territory; and Texas has always had its own land-laws, agricultural and mineral, without regard or subordination to those of the United States. In other words, the United States, not being the *owner* of lands in Texas, has not asserted, on the ground of sovereignty, any right to the ownership or control of the agricultural, mineral, or dynamic (water-power) resources of that State. Since any proposition putting the States of the Union on any other basis, and determining on historical grounds the sovereign rights of each, would be absurd, we may assume that all the States possess the same sovereign rights as Texas, and that the Federal government can constitutionally control within their borders only the administration of its own property.

This limit has been, in fact, carefully observed hitherto in Federal legislation. The U. S. "mining-laws" are not, strictly speaking, mining-laws at all. They are simply statutes regulating the disposal of mineral lands belonging to the Federal government; and they leave to local authority the regulation of the industry of mining itself. All put together, they affect nothing but the title derived by the miner, as tenant or purchaser, from the United States as owner.

With regard to the mineral resources owned by it in existing States, or in Territories which are destined to become States, the policy of our government hitherto has been, since 1866, to treat such natural resources as the prospective property of the citizens of the States within which they are located, subject to the regulation, control, and taxation by those States; and the lands containing such resources have been either conveyed outright to the States concerned, or sold to individuals, to be thereafter controlled, as to their development, by the local governments.

The first public domain of the United States was acquired by cessions from some of the original thirteen States, covering what is now Tennessee, Illinois, Indiana, Ohio, Michigan, Wisconsin, that part of Alabama and Mississippi lying north of the 31st degree of North Latitude, and that part of Minnesota lying east of the Mississippi. Nearly all the public land in Tennessee was absorbed by the claims of North Carolina, and the remainder was subsequently ceded by the United States to Tennessee.

The rest of the territory granted by the States to the Federal government included our first public domain, together with lands already owned by private persons.

The first Federal legislation concerning the mineral resources of the public domain is found in the famous ordinance of 1787, concerning the Northwest Territory (comprising what is now Ohio, Illinois, Indiana, Michigan, and Wisconsin), which provided that one-seventh of the land should be given to the soldiers of the Continental Army, one-seventh reserved to the United States, one-thirty-sixth set apart for the support of schools, and the rest allotted to the States "*pro rata*," and sold at auction by the State Commissioners, who were to give deeds, excepting and reserving to the United States one-third of all gold-, silver-, lead-, and copper-mines. The same reservation applied to the bounty- and school-lands also.

I do not propose to trace here the further course of this legislation, or the history of the Northwest Territory, out of which Ohio was carved in 1800, Michigan Territory in 1805, and Illinois Territory in 1809, leaving (under the name of the Indiana Territory) what became in 1816 the State of Indiana. Nor is it necessary to discuss here the Louisiana purchase of 1803, or the question whether Idaho, Washington, and Oregon were acquired by that bargain, or belong to the United States by virtue of the discovery of the Columbia river by Capt. Gray (1792), the exploration of Lewis and Clarke (1805), the settlement of Astoria (1811), or the cession of "the Floridas" by Spain (1819).

In 1807, Congress enacted that the lead-mines in the Indiana Territory (then comprising Indiana and Illinois) should be "reserved for the future disposal of the United States;" that any grant by the United States of a tract in which a lead-mine had been previously discovered should be considered fraudulent and void, and that the President was authorized to lease for not more than five years any lead-mine which had been or might thereafter be discovered in the said Territory.

This act inaugurated the policy of leasing the public mineral lands. The leases executed under it covered tracts, at first 3 miles and afterwards 1 mile square, and required the working of the mines with due diligence, and the return to the United States of 6 per cent. of the ore mined.

This policy was continued for 40 years, uninterrupted by any further legislation, except the Act of June 26, 1834, creating an additional land-district in Wisconsin, which authorized the President to offer for sale certain public lands, "any law of Congress heretofore existing to the contrary notwithstanding." Many Acts between 1807 and 1834 had dealt with the pre-emption and disposal of public lands; but nearly all, if not all, of them had continued the reservation of 1807 as to lead-mines. But this Act of 1834, expressly nullifying previous reservations, seemed to permit the pre-emption and acquisition of the lands which it specified, whether they contained lead-mines or not. At all events, this was the plea of the defendant in the case of U. S. *vs.* Gear.[1] In that case, decided in 1845, the U. S. Supreme Court held defendant guilty of trespass, on the ground that the reservation not expressly contained in the Act of 1834 was to be regarded as a part of it, because it had been for many years the consistent policy of the United States, and could not be set aside by inference from the terms of a law, the declared purpose of which was simply the creation of a new land-district. In a previous case,[2] the Court had said in 1840:

> "It has been the policy of the government, at all times, in disposing of public lands, to reserve the mines for the use of the United States."

The decision in the Gear case was reaffirmed in Cotton *vs.* U. S. (11 Howard, 229), which applied the same principle to the cutting of timber upon public land, declaring that, as the owner of the land, the government had the same rights as an individual owner to protect its property.

The constitutionality and equity of these decisions cannot be seriously questioned. They rest upon the ownership by the United States of the lands concerned, and therefore upon the common-law right of a holder of the complete title—not upon the prerogatives of a sovereign. Yet there is, as Curtis H. Lindley says, no doubt that the reservation of part of the product of certain kinds of mines—for instance, in the Ordinance of 1787, above cited—was suggested by the similar provisions in the Crown grants, upon which the titles of the original States were based. We do not find, however, that these *regalia*, or rights of the sovereign, as distinguished from the rights of the land-owner, were ever seriously enforced. The policy of the government, so plainly adopted in 1787 and 1807, was applied to its own lands only.

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[1] 3 Howard, 120.

[2] U. S. *vs.* Gratiot, 14 Peters, 538.

It is interesting to note in passing that the doctrine of sovereign rights over mines has been at least twice asserted by single States against the United States, and not without some appearance of logical force. For, if the United States, being at once a land-owner and a sovereign, does not possess, or has waived or ceded, its claims in the latter capacity, then, in the former capacity it can be dealt with by the sovereign State, like any other private land-owner. Thus it was held, in 1853, by the courts of California,[3] that the State, by virtue of its sovereignty, enjoyed all the *jura regalia* belonging, under the common-law, to the King of England. The doctrine appears to have been that, whoever owned the land containing the mines, the Sovereign only could control their operation. In the case of California, such a sovereignty could not have come from the King of England and must have come through the King of Spain and his assign, the Republic of Mexico, the rights of which had passed, through conquest, to the United States. The only ground of California's claim must therefore have been, that by its admission to the Union it became a sovereign State, clothed (from whatever source) with the rights enjoyed by the original thirteen. Whatever merit this claim may have had, it was subsequently abandoned by the California courts.[4]

But a similar doctrine was apparently followed by the Legislature of Michigan, in its Act of April 28, 1846, which declared that property in all mines of gold or silver, or of other minerals containing gold in any proportion, is vested in the people of the State of Michigan by virtue of their right of sovereignty. Since the Federal land-laws applied to public lands in Michigan until 1873, this assertion of "sovereignty" over certain mines, if extended over such lands, could be based only on the theory that the sovereign rights of the original States which ceded to the United States the territory out of which the State of

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[3] Hicks *vs.* Bell, 3 Cal., 219. I take this statement and reference from Curtis H. Lindley's admirable and almost exhaustive treatise on American Mining Law, 2d ed., San Francisco, vol. i., Sec. 21, p. 42 (1903).

[4] Moore *vs.* Snow (1861), 17 Cal., 199, 217; and later cases cited by Lindley.

Michigan was subsequently segregated were somehow held in trust by the United States, and afterwards tacitly re-conveyed to the State at the time of its admission to the Union. No conflict having arisen under this claim between the State and the Federal authorities, it may be regarded as practically settled that neither party now asserts "sovereign," as distinguished from proprietary, rights to mines.

The policy established in 1807, of reserving from sale, and holding and leasing, the public mineral lands was, as I have said, maintained for 40 years—a considerable period, in view of the total length of our country's history.

In December, 1845, President Polk, in his first annual message to Congress, reported that the results of the leasing of mineral lands had been in every respect unsatisfactory. The revenue to the government had been less than one-fourth the expense of administration; litigation and wasteful mining had been promoted, etc.; and he recommended the abandonment of the system. In 1847, this recommendation was adopted by Congress, and the long experiment of leasing public mineral lands came to an end—not to be revived until, after more than 60 years, it was proposed as a new idea by a generation which had forgotten the experience of the fathers.

The discovery of gold in California, and the consequent rush of many thousand adventurers into the unsettled and unsurveyed territory acquired from Mexico by the treaty of Guadalupe Hidalgo, put a new face upon the question of Federal ownership and sovereignty. The pioneers who thronged to the gold-bearing gulches not covered by any asserted private ownership were clearly, according to the previous decisions of our Supreme Court, trespassers upon the property of the United States; and, in all conflicts of title among them, the superior title of the United States would theoretically operate as a bar to such "equity" remedies as writs of injunction or ejectment. But the greater part of the disputes over mining-titles in those regions involved just such remedies; and it was difficult to see, where both parties were trespassers, how one could ask protection against the other.

The courts solved this difficulty in a summary and sensible way, by leaving out all consideration of the superior title of the United States, and adjudicating the relative rights of possessory claimants to public mineral land, with the tacit proviso that the rights thus declared were subject to any future action on the part of the real owner. Under this *modus vivendi*, the treasures of the Comstock lode and of many other rich deposits were claimed, fought for, and won by individuals or corporations which had no original title to them, but only, according to local regulations, a better claim than their antagonists. Yet these regulations were made by communities of trespassers, without regard to the superior authority of the Federal landlord.

This curious situation lasted until 1866, when Congress passed an Act, declaring the occupancy and exploration of the public mineral lands free to all citizens, and confirming the rights which had been acquired in these lands under local regulations. In other words, the United States ratified whatever had been legally done without its assent, and agreed that its silence should be, within certain limits, interpreted as consent. This Act was confessedly crude and imperfect, and it was found to be particularly objectionable, in that it did not provide, either through possessory occupation and development, or through purchase and patent, for the protection of the occupant of a piece of mineral land (whether as tenant or as grantee of the United States) against predatory explorations and exploitations of his location. It was "the lode" only, with its "dips, spurs, and angles," which was granted under this law. And, since the law recognized the conditions of title imposed by "local regulations," and the one universal condition of such regulations was priority of discovery and location, this became naturally the basis of possessory title.

It was not until 1872 that "The Law of the Apex" came. This was a crude attempt to remedy the defect of the Act of 1866, by protecting the occupants of mining-claims against piratical trespass, through the grant to them of a right to the land, as well as the "extralateral" right. Unfortunately, these rights were made dependent upon a new and previously unknown condition—the "apex"—and subject to the similar rights of any neighboring "apex."

It is unnecessary to dwell here on the positive and negative defects of this law. It was a great mistake to make mining-titles dependent upon future geological developments. The statute created greater evils than those it was designed to cure. And, in my judgment, it will never be too late to repeal it, and to substitute the simple and logical system followed by all other nations and by the majority of the States of the Union.

Nevertheless, the Acts of 1866 and 1872, however defective and mischievous they may have been, possessed one feature, which their faults did not nullify: namely, they invited the exploration and occupation of the mineral public lands, and thereby stimulated a development unparalleled for rapidity and extent. This result would have been equally remarkable, and still more satisfactory, if the liberal policy of the United States had been so exercised as to give safe and definite titles to the occupants and purchasers of its mineral lands. The circumstance that a tenant or grantee of the government was liable to an interminable risk of losing title, by reason of geological developments concerning the "apex" of his lode, would not seem, at first glance, to be an additional encouragement to his enterprise. Still, it must be confessed that, during the period between 1866 and 1872, and for some years after the latter date, the "extralateral right" attached to a mining-location was so attractive to prospectors and possessory owners as to counterbalance the two great statutory evils—namely, during the first period, the total lack of protection of the claim-owner against trespass upon his claim, and during the second period, the uncertainty of title caused by basing all the locator's rights upon the location and course of the "apex" of the located lode, without regard to the first discovery thereof. The positive grant of the "extralateral right" loomed large in the eyes of the prospector and his assignees; while the precarious conditions of that grant, and the express reservation of the corresponding "extralateral right" of a neighbor,—raising questions which could only be settled by litigation, future development, expert evidence, etc., during which the United States would simply stand back, and let its grantees fight out for themselves the exact meaning of its grant,—were not realized at once as real defects of title. These defects in our mining-law have cost American citizens many million dollars. Yet, in spite of this huge, unnecessary handicap (imposed without the least benefit to the government), the policy of transferring the mineral public land to individual ownership upon terms so liberal as to amount practically to a gift, has resulted in the creation of a new empire, west of the Rocky mountains, with a rapidity unprecedented in human history. I think no one would venture to dispute the proposition, that, through its homestead-law, its mineral-land laws, and its grants in aid of transcontinental railroads, the United States has built up a new, rich, loyal Western realm, which could never (or only after much longer processes) have been established in any other way.

It is now proposed to abandon the policy which has produced such results, and to revert to the plan which so conspicuously failed a century ago. The reasons for that failure still exist. The causes which produced confusion, corruption, and waste before are even more likely to produce them to-day. It is not easier to administer the mineral deposits of Alaska than it was to manage those of Illinois. It remains as true as ever, that the wealth, prosperity, and loyal union of our States is best served by placing their natural resources, as rapidly as possible, in the hands of their citizens, subject to local taxation and regulation. We have discovered no new principle in democracy which nullifies Spencer's famous declaration that a democratic is the worst form of government for doing those things which no government ought to do.

The following propositions concerning the proposed policy are submitted with considerable confidence:

1. It is never desirable, even though it be sometimes necessary, that private enterprise should be subject to governmental interference or competition.

2. If any government in this country must thus interfere, it should be the local (municipal or State), and not the Federal, government. Among the many arguments for this proposition I may mention a few.

a. As a matter of statesmanship, the Federal government cannot afford to put one State (or Territory destined to become a State) on a different basis from others; and since it cannot legally administer the natural resources of New York or Texas, the attempt to do so in Idaho or Alaska would be unwise. This point I have already discussed.

b. It is a well-known teaching of experience that the government dealing with local interests and industries should be, if possible, a local government, easily watched and reached by the citizens affected. In my discussion of Mr. Bain's paper on Alaska Coal-Land Problems,[5] I have emphasized this point, and will not here repeat what I said there. But I may add that a central government attempting to perform such local supervision invariably fails to do that work well, and incidentally falls short of vigilance and efficiency in the functions more properly belonging to it. Take, for instance, the administration by the British Parliament of the internal affairs of Ireland, Scotland, Wales, and England, as well as India. The case is plain: Parliament has too much to do, and consequently delegates to committees and departments the work for which it is nominally responsible. Another instance may be found in the government by Congress of the District of Columbia, which receives perfunctory attention at a couple of sessions during the whole term. The necessary result is that the affairs of the District do not receive the vigilant attention and public discussion which they would get from a District legislature, not burdened with the management of national, international, and colonial questions, and the manufacture of party ammunition for presidential campaigns. Moreover, the actual and necessary business of the national government is neglected by those executive departments which are loaded with extra-constitutional duties. Administrations not seldom claim much credit for the fearless exposure of long-continued department abuses which, in fact, they could have prevented if they had not been so much occupied with other people's business that they neglected their own.

3. Of all the methods of governmental administration of natural resources, that of leasing, as now widely advocated, is the worst. I refer to the system of short leases (the term usually mentioned being 25 years), coupled with the requirement of continuous operation. I could ask no better illustration than the history of the Pennsylvania anthracite-region. In his paper on Alaska Coal-Land Problems,[6] Mr. Bain gives a summary of this history, which candidly confesses the beneficial effects wrought by the large corporations, so often ignorantly denounced. But he omits to mention the greatest cause of the

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[5] *Bulletin* No. 68, August, 1912, pp. 811 to 842.

[6] *Idem.*

confusion and waste which those corporations reformed, namely, the system of short leases prevalent throughout the anthracite-region. No amount of stipulation, regulation, and inspection can make it the interest of a lessee to spend money on improvements for the benefit of his successor; to put up, at extra cost, machinery and buildings for permanent use, after his term; or to sacrifice immediate profit in favor of technical efficiency and economy. And no amount of governmental pressure can be reasonably or practically applied which will make a mine-tenant do as much for conservation, scientific improvement, etc., as he would voluntarily do if he were the owner of the mine.

This is peculiarly true of coal-mines. I notice that the advocates of a leasing-system always propose to bind the lessee to operate continuously; yet this condition is fatal to economy. Mr. Parker, who has observed the evils of hot competition in the bituminous coal-mines, thinks the government should interfere by ordering those mines to be shut down which do not pay![7] But why do they go on, if they do not pay? Doubtless because their owners would rather sacrifice some money for a while than incur a greater loss by stopping short. All sorts of reasons make a man go on for a while in a losing business; and operators possessing large capital can often afford to do so, for the sake of a more profitable future. But to make the title to the property depend on its continuous operation is to put large and small capital into common ruin, and to ordain deliberately the waste of natural resources. If government leases were granted for very long terms, and (like many private mine-leases) could be held by annual payments during periods when the mine was not worked, this objection would be measurably met. But that would be practically granting ownership of the mines—which is what wisdom and experience recommend, and not what our reformers want.

4. Why do we not hear of the leasing of gold-, silver-, and copper-mines, and of agricultural lands? The argument for these applications of the system would be equally strong. I fear the reason for its conspicuous absence is, that coal-mines require capital, and that capital is an easy mark for destructive attack. But, as has often been observed, capital does not fight, except when cornered. As a rule, it runs away.

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[7] The Conservation of Coal in the United States, *Trans.* xl., 601 (1909).

This, I think, will be the probable end of the cycle into which we are entering. If we try this new system, which is not new, we shall find, as was found three generations ago, when it *was* new, that it is not good; and we may then be content to try once more the old-fashioned road on which our nation traveled to greatness. The men who laid out that road "did not foresee," we are often told, the development of the vast region of the future through which it was to lead. But they were skillful surveyors; they knew where the road should be placed;—and historians tell us that roads are the most permanent works of man!

SUBJECT TO REVISION.

[TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS.]

The Effect of Alumina in Blast-Furnace Slags.*

BY J. E. JOHNSON, JR., ASHLAND, WIS.

(Cleveland Meeting, October, 1912.)

THE subject of blast-furnace slag is one which has had much consideration, particularly from the scientific standpoint, and several years ago technical literature contained many learned discussions on the oxygen-ratio of the acids and the bases, all on the assumption that acids and bases must be present in a ratio depending on molecular weight and chemical valency. Much work has also been done, sometimes in conjunction with the former, on the total heat and the temperature of initial fusion or softening of various slags.

A wider knowledge of the blast-furnace now enables us to say that for practical purposes this work has been of almost no value. It is now generally admitted that on the physical side the important point of a given slag is neither its total heat of fusion nor its softening-point, but its free-running temperature, since this is the critical temperature of the furnace.

This temperature bears no definite relation to the softening-temperature, as determined by Seger cones, for instance, because some slags melt relatively like ice, with a very short viscous-range, while others melt relatively like tar, with a very long viscous-range.

In one the difference between the softening- and the free-running temperature may be very small, as little as 100° F., while in another this difference may easily be 200° or 300°.

The importance of this difference is so great in practice as to destroy completely the value of exhaustive investigations, even such as those of O. Boudouard, published a few years ago.[1]

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* Presented also at the Joint Meeting of the Institute with Section IIIa, Metallurgy and Mining, of the Eighth International Congress of Applied Chemistry, New York, September, 1912.

[1] It may not be amiss at this point to call attention to the excellent suggestion of Woolsey McA. Johnson, made several years ago, for determining the temperature at which different slags have the same practical fusibility. This suggestion

When we come to the chemical side of the question we may as well dismiss the obscuration which comes from the introduction of oxygen-ratios and other abstruse theoretical considerations, and admit frankly that experiment on the furnace itself is the only safe foundation for practice or for a useful theory.

I am moved to lay stress on this point because in years past I was greatly puzzled and befogged by some of the older school of furnace-men, who emphasized the mysterious nature and difficulty of making slag-calculations, the fact being that, given the materials to be used and the slag to be produced, the college-student who could not be taught in a day to make the calculations, would have a poor chance of ultimate usefulness. The omission or concealment of the fact that the slag to be produced was known only by experiment and experience with the furnace itself, caused my mystification, and I therefore emphasize it here.

The object to be sought may be briefly stated as follows:

Given the materials to be used and the kind of iron to be made, to ascertain the slag which will produce the result with the least cost for coke and for flux, and will permit the greatest output. The three desiderata come in the order given.

We may in this discussion omit consideration of charcoal-practice because the slag-question is of minimum importance in that field, and the field itself is of minimum importance in the iron industry. Also because of my complete ignorance of that practice.

(This paper was written three years ago, but has remained unpublished until now. It so happens that in the interval I

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was briefly to make experiments in a small electrically-heated furnace, the bottom of which was a carbon slab with a small round hole through it. The temperature at which different slags would flow through this hole would furnish accurate information as to their relative practical fusibility.

To determine the proper size of the hole through the carbon block, I offer the further suggestion that the temperature of the slag flowing over the cinder-dam in standard furnace-practice be taken when the slag was about of normal composition. Then put some of this slag in the experimental furnace and heat it to the temperature observed. A few trials would determine the size of the hole through which it would flow at this temperature. This size of hole could then be used in future experiments with the certainty of obtaining results comparable with actual practice.

Investigations along these lines would be of real interest and value to furnace-men.

have had extensive experience with charcoal-iron slags, but it is not considered desirable to introduce a consideration of that subject into this paper, because in the absence of the necessity for desulphurization, the problems offered by charcoal-slags are totally different from those of coke-slags. In a general way, however, it may not be amiss to state that observation of charcoal-slags has confirmed the position taken in this paper.)

Coming then to normal coke practice and remembering that generally the lowest fuel-consumption is obtained with the lowest critical temperature, the object desired may be stated more definitely as the production of the most fusible slag that will give the necessary desulphurization of the iron. This statement is perhaps subject to certain limitations from the fact that there are circumstances in which it is necessary to raise the critical temperature in order to enable the iron to absorb a large quantity of silicon. The extent of this limitation I am not able to state, but we will return to this subject later.

It is necessary to remember that while the effect of lime is to increase the basicity of slag and facilitate the removal of sulphur, its use is subject to grave limitations, for two reasons:

1. The lime being generally the major ingredient of the slag, to increase the percentage of the lime in the slag involves a more than proportional increase in the slag-volume; for instance, to increase the percentage from 50 to 55 requires an increase of more than 10 per cent. of the weight of the slag in the lime, or about twice as much in limestone.

2 (and much more important). The addition of lime raises the fusion-point of the slag very rapidly.

The desulphurizing effect of the slag is proportional, not only to its basicity, but also to its fluidity in an almost equal degree, so that while increased lime *per se* has a desulphurizing influence, this is, to an increasing extent, neutralized and finally reversed completely by its decreased physical activity.

This is well shown in basic practice, in which the highest sulphur iron is made, not with deficient lime, but with an excess (due to a change in ore or the like) so great that the heat available is unable to bring the very refractory slag to the free-running condition.

In further illustration of the point, there are two distinct methods of making this kind of iron. The first consists in running on a very calcareous slag, with which the silicon in the iron is kept down by the basicity of the slag in spite of the high temperature necessary to keep the latter fluid. The second consists in maintaining a slag of only moderate basicity and much lower fusion-temperature and keeping down the silicon by carrying a heavy ore-burden, which, of course, can easily be done with the lower critical temperature.

Furnaces running on the first plan always require more coke for basic than for foundry-iron, while those running on the second plan use less than for foundry-iron.

It seems to me practically certain, therefore, that there is a considerable range of lime-content in the slag for given conditions, in which the desulphurization of the iron is not appreciably affected, while the coke-consumption necessarily rises with the increase in the fusion-temperature consequent on the higher lime-ratio, as well as the increased slag-volume.

In the case of foundry-iron, the increase of lime may be positively detrimental because of the reduction of silicon in the iron. I have recently been informed of several cases in which the lime had habitually been carried too high, because this action was not understood. These cases of bad furnace-operation show the importance of this subject.

Effect of Magnesia.

In all theoretical slag-calculations the quantity of magnesia is multiplied by 1.4 to put it on the same basis as the lime, which, from the point of view of the oxygen-ratio and molecular weights, is perfectly correct. Practically, however, this does not work out. Furnace-men using a limestone with a variable content of magnesia have told me that careful observation had failed to show any difference whatever on the furnace whether the magnesia was high or low, and I have seen a furnace using half calcite and half dolomite, the calcite being relative impure and the dolomite very pure, put onto all dolomite, which should have made the slag excessively basic, on the basis of molecular weight, but which as a matter of fact showed no observable change.

The truth seems to be that magnesia is less active chemically than lime in about the same proportion that its molecular weight is less.

An interesting proof of its small chemical activity is supplied by the manufacture of caustic soda from the carbonate. Quicklime is added to the carbonate solution and takes up the carbon dioxide from the soda; for this purpose magnesia is found to be perfectly inert and worthless. Sir Lowthian Bell's opinion that magnesia was so inert as to be useless for the removal of sulphur in the blast-furnace is well known, but this opinion is no more borne out by practice than that giving it a greater value than lime, the truth appearing to be that for furnace purposes, in all ordinary proportions, one is about as effective as the other.

It is very necessary, however, to recognize that the addition of a certain amount of magnesia has a marked effect in lowering the fusion-temperature of the slag, and is therefore of great use where calcareous slags are required, particularly in the manufacture of basic iron.

For practical purposes lime and magnesia may be considered as being of equal value, and hereafter in this paper "lime" will be used to mean the sum of lime and magnesia.

The Effect of Alumina.

The foregoing portion of this paper contains nothing essentially new and is intended as an introduction to the remaining portion, the substance of which seems to have been unknown to most of the furnace-men with whom I have discussed the subject.

The effect of alumina has been the subject of much discussion; some regard it as an acid, others as a base, while a few declare it can be made to act as a base or an acid almost at will.

It has seemed to me that under such circumstances the probability was that its action was neither acid or basic, but was perfectly neutral, simply a diluent affecting the viscosity of the slag to some extent, but, with a given ratio of lime to silica, not affecting its chemical nature at all.[2]

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[2] This conception of a given constituent affecting the physical but not the chemical properties of slag, in which two kinds of properties are so closely interwoven, is difficult at first sight, but I have illustrated it for myself as follows. If we had an ordinary acid, such as hydrochloric acid, in one beaker, and a solution of caustic alkali in another, we could mix the two in any proportions and each addition of one or the other would correspond to the change in the acidity or basicity of the resultant solution, but if we added a considerable quantity of molasses, we should alter its viscosity and its "free-running temperature," without affecting its chemical properties in any way. The illustration is a homely one, but in my opinion the case is precisely parallel to that of slags.

For several years the range of alumina in slag of which I had knowledge was so limited that I could not prove this contention, but very recently a furnace-man gave me complete information of a remarkable series of experiments he had carried out, in which the alumina in the slag had been as high as 39.5 per cent. with silica as low as 21 per cent. on individual flushes, and averaging for an entire day SiO_2, 24.7; Al_2O_3, 36.0 per cent.

The iron made was good Bessemer iron, about 2 per cent. of silicon, with sulphur about 0.023 per cent.

With this let us compare standard Lake-ore practice on basic iron, running about Al_2O_3, 13.5; SiO_2, 33 per cent.; and Virginia practice on the same iron, Al_2O_3, 6.5; SiO_2, 36 per cent.

It is unfortunate that the experimental run was on Bessemer instead of basic iron. The records of other days which show basic iron made are not so complete, but they indicate only a small difference in the slag; that shown, of course, being an increase in lime.

The coke-consumption of this slag was not materially different from what it was in standard practice; the slags were free-flowing, and did not have a noticeably higher fusion-temperature than ordinary.

Here, then, we have three cases, in all of which the coke is of about the same sulphur-content, the desulphurization of the iron is the same, the coke-consumption is no more different than would be accounted for by the different kinds of ore. The only difference of importance is the silicon in the iron, which is not sufficient to require a very great change in the slag-composition. We will refer to this condition later.

The amount of lime in these slags can be calculated, but it is necessary to remember that there is a small quantity of neutral material, CaS, FeO, MnO, etc., which may be taken at 3.5 per cent. in all cases as a close approximation to good practice. Following this procedure, the results shown in the first four columns of Table I. are obtained:

TABLE I.—*Composition of Slags.*

	1. Al_2O_3 Per Cent.	2. SiO_2 Per Cent.	3. CaO by Difference. Per Cent.	4. Neutral Substances. Per Cent.	5. Ratio $\frac{CaO}{Al_2O_3+SiO_2}$	6. Ratio $\frac{CaO+Al_2O_3}{SiO_2}$	7. Ratio $\frac{CaO}{SiO_2}$
Virginia practice..	6.5	36.0	54.0	3.5	1.27	1.68	1.50
Lake-ore practice..	13.5	33.0	50.0	3.5	1.08	1.92	1.51
Special experiment	36 0	24.7	36.8	3.5	0.59	2.90	1.44

Columns 5 and 6 present the ratios of lime to silica + alumina, and lime + alumina to silica. These ratios change from 1.27 to 0.59 in the first case, and from 1.68 to 2.90 in the second case.

No one can hope to show any relations between these ratios that can bear in any intelligible way on the variation of the alumina-content.

Column 7, however, shows the ratio of the lime to the silica, which is virtually a constant throughout. The result in the experiment is the lowest, and corresponds to the higher silicon in the iron.

If the lime were increased in this latter case about 5 per cent. of its own weight, this slag would then have identically the same ratio as the others, and would be just about as much "limier" as would permit the production of basic iron by the addition of a little more burden.

It may be claimed in opposition that the analyses chosen as representative of Virginia and Lake-ore practice have been taken with the object of showing the result desired rather than representing the typical slags. For the Virginia practice this certainly is not true, and if there be any error in the slag-analysis for Lake-ore, I do not know how to better it. Certainly no correction that could be made would fail to leave the lime-silica ratio infinitely nearer a constant than either of the others.

It may be well to reiterate here that such a comparison is only useful in the case of furnaces on a comparable basis in other respects, that is to say, working for the same degree of desulphurization and the same silicon, and using materials of about the same sulphur-content.

The question of the slag-volume also enters here, and the question whether a given slag-volume is as efficacious in the removal of sulphur when the alumina is high as when it is low. We know that if the slag exceeds a certain content of sulphur in ordinary practice, the desulphurization of the iron will be incomplete, and if the alumina be simply a diluent, the effectiveness of the slag might be diminished in proportion as the percentage of the alumina rose.

In spite of this theoretical consideration, the sulphur in the slags in the high-alumina experiments mentioned ran about

1.75 per cent., with perfectly satisfactory desulphurization, while the relative slag-volume was little, if any, higher with alumina approaching 40 per cent. than with normal alumina.

On foundry-irons the ratio of lime to silica is, of course, lower than on basic. In Virginia practice the typical analysis would be about: Al_2O_3, 7; SiO_2, 41; CaO, 48.5; neutral substances, 3.5 per cent., giving a lime-silica ratio of 1.2, instead of 1.5 on basic iron.

A recent opportunity of discussing this subject with many furnace-men in all parts of the country has shown that virtually all slags for foundry-iron have a lime-silica ratio of between 1.2 and 1.4.

Foundry-iron practice is so much less definite than basic that the same exactness of ratio is hardly to be expected, and data are not at hand to determine a correct ratio definitely.

There seems to be a fairly well-defined opinion among the best informed furnace-men that alumina has the effect of making the slag viscous and stringy in a certain range, beginning about 16, reaching a maximum about 20, and disappearing between 25 and 30 per cent.

The data at hand do not permit of a definite statement on the subject, because the lime-silica ratio is not known for a sufficient number of cases, and if high alumina be coupled with a higher than normal lime-silica ratio, it is not fair to blame the alumina any more than the high lime for the result.

One case is on record in which the high alumina was coupled with excessively bad results, but it was also coupled with an extremely high-sulphur coke, which compelled the use of very high lime, which resulted in an infusible slag. In such a case the high alumina may do no more than render stringy a slag that would be troublesome in any event.

A question of this character can only be properly settled by the use of an accurate pyrometer for determining the temperature at which different slags become free-running. The perfection of an accurate optical pyrometer which can easily be used to determine the temperature of the slag close to the furnace would make the investigation relatively easy.

The utility of such knowledge lies in enabling us to determine in advance the slag which will give satisfactory results in practice when a change in the materials used causes the alumina to exceed the ordinary limits.

The decreasing grade of the ores available must result in the increasing use of many ores higher in alumina than those of present practice. If the laws governing its action are not known, bad results are bound to follow.

A furnace-man recently told me of a case in which the alumina had been regarded as an acid, and the slag had been calculated to give the same ratio of lime to silica + alumina as in standard practice, with the result that the furnace had been badly "bunged up" with lime.

Several furnace-men with whom I have talked had realized that alumina could not be properly regarded in this way, but only two have made definite statements that they regarded it as a neutral substance, and only one had reached the conclusion that a definite lime-silica ratio was the object to be attained. This was J. H. Frantz, General Manager of the Columbus Iron & Steel Co., whom I first met when this article was almost complete. Several years ago he had reached the opinion here maintained, and had prepared a table giving a lime-silica ratio of 1.5 with percentages of alumina varying from 8 to 30 per cent. He also told me that this view had been published several years ago, but he did not remember the name of the publication or of the author.

The fact that this publication as well as the principle seems to be entirely unknown to most furnace-men, and the increased importance attributed to the free-running temperature of the slag in recent years, are the only excuse for the present paper, in the face of prior publication of similar matter.

In conclusion, it seems worth while to point out that the increased viscosity accompanying high alumina may be of benefit in making foundry-iron. This was a subject to which both Mr. Frantz and I independently had given some consideration for several years.

That iron will not take up much silicon except at a relatively high temperature is well known, and as its melting-temperature is relatively low, its tendency is to run down out of the region in which it can absorb silicon into the crucible.

The fusion-temperature of slag is much higher than that of iron, and being more viscous, it acts as a retardant on the iron and delays its descent so that it can acquire the necessary temperature.

A too-fusible slag does not perform this function properly, and I have personally seen a case in which no reasonable reduction of burden would raise the silicon to ordinary foundry-limits, but in which this was easily done by increasing the lime considerably beyond that necessary for desulphurization, in spite of the desiliconizing influence of the more basic slag. For the same reason it is more difficult to make foundry-iron when enough magnesia is present to increase the fluidity of the slag.

Now if alumina increases the viscosity of the slag without the accompanying desiliconizing influence of the limier slags of the same free-running temperature, then the introduction of the silicon into the iron should be facilitated by high alumina.

Mr. Frantz is of the opinion that this is true to some extent. My own experience does not cover the point.

SUBJECT TO REVISION.

[TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS.]

Fires in Metalliferous Mines.

BY GEORGE J. YOUNG,* RENO, NEV.

(Cleveland Meeting, October, 1912.)

I. GENERAL.

THE recurrence of mine-fires in Nevada during the past decade is not only a matter of interest, but also one of considerable concern to engineers and mine-managers. The more important fires may be enumerated as follows:

Forman Shaft fire, Gold Hill, April 21, 1903; shaft-house, machinery, and shaft destroyed; loss estimated at $50,000; cause unknown.

Union Shaft fire, Virginia City, July 14, 1904; shaft-house, machinery, and shaft in part destroyed; loss estimated at $100,000; cause of fire stated as the careless throwing of a match in the rope-house.

Sutro Tunnel fire, Virginia City, Jan. 27, 1909; 700 ft. of tunnel-timbering destroyed, and direct damage of $10,000; cause "probably electric wires."

Belcher Shaft fire, Gold Hill, Aug. 9, 1910; shaft-house destroyed, machinery ruined, and upper part of shaft damaged; damage, $25,000; no cause given.

Belmont Mine fire, Tonopah, Feb. 23, 1911; fire started in winze; 17 men killed.

Giroux Mine fire, Kimberly, Aug. 23, 1911; fire originated in shaft-station from unknown cause; 7 men killed, and $34,521 damage caused.

(NOTE 1.—On June 11, 1912, some time after the present paper was written, a fire broke out in the pump-room on the 2475 station of the Ward shaft, Virginia City. This fire was caused by the short-circuiting and exploding of the starting-switch on No. 5 pump. The oil in the switch was scattered about and set fire to the timbers and lagging of the pump-room. The pump-man in charge notified the surface, and the electric current was shut off. A hose was then turned on the fire, but the dense smoke drove the attendant out. Inspection by the electrician and the shaft-foreman, equipped with oxygen-helmets, soon after, showed no flames or excessive heat, but the continued expulsion of smoke indicated that some fire was present. The pump-chamber was then ordered flooded. No estimate of the damage was published and no fatalities occurred.)

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It is not the purpose of the present paper to discuss the above examples, but rather to use certain features of them to formulate general plans which might be of use in fire-prevention and fire-fighting.

In the above list three fires are noteworthy in that they originated in wooden shaft-houses and communicated with workings underground, resulting in considerable damage. The risk taken by closing in the mouth of a shaft with wooden buildings has long been recognized, and most of the Western mines have eliminated the shaft-house. In Virginia City, on account of the severe winters, the shaft-house has been retained, but, following the lesson given by these three fires, and the passage of a mining-regulation governing this feature of mine-construction, active steps have been taken to remove the risk. In almost every case where the shaft is used as a working-shaft wooden buildings about the shaft-mouth have been removed.

Mine-fires may be considered in two groups: those which occur in the surface-plant and those which occur in the underground workings. Fires in mine surface-plants do not, as a rule, result in loss of life, and if proper fire-fighting facilities are provided, the fire may be extinguished with but moderate loss. In Nevada considerable progress in the use of fire-proof construction has taken place in recent years. The following steel and semi-fire-proof constructions may be noted: Goldfield Consolidated mill, Goldfield; New Belmont mill, Tonopah; Nevada Consolidated concentrating-mill, McGill; Nevada Reduction Works mill, Dayton; Nevada Hills mill, Fairview; Pittsburgh Silver Peak, Blair.

Steel head-frames: Montana Tonopah, Tonopah; Tonopah of Nevada, Tonopah; Union shaft, Virginia City; Star Pointer shaft, Ely; Merger mines shaft, Goldfield; Giroux mine, Kimberly.

The protection of the surface-plant from fire is recognized as a necessity, and most mining companies, where an extensive plant has been installed, provide facilities of some sort for fire-fighting. In the design of a surface-plant the question of fire-risk should receive considerable attention. The segregation of different parts of the plant, the separation of each unit by a sufficient distance to prevent the spread of a fire from one unit to another, and, finally, the use of materials which are either fire-proof or of a slow-burning nature, are the main points which deserve consideration at the start. While it is true that the use of expensive materials and equipment is not warranted in the early stages of a mine's development, it is often the case that after a mine has reached the producing stage the same types and materials of construction are used, and a surface-plant grows until it involves an amount of combustible material that in itself is a risk of some magnitude. The inevitable happens, a fire gets beyond control, and the destruction of the plant follows; this occurs in spite of what were deemed adequate fire-fighting facilities.

That a more general use of fire-proof and semi-fire-proof materials and construction in Western mines is merited goes without saying. That there is progress in this direction no one familiar with the more recent camps of the West can deny. It may not be out of place here to review the different types of building-construction used for surface-plants. They may be enumerated:

1. Timber frame with board siding.
2. Timber frame with corrugated-iron siding and roof.
3. Steel frame with curtain-wall construction in which brick, ferroinclave, and cement plaster or reinforced concrete is used.
4. Reinforced concrete.
5. Brick walls, steel frame, and corrugated-iron roofing.

In Nevada the first two methods of construction are common in the case of most mines and prospects. Of these, the second method is preferable on account of reducing the amount of combustible material. By whitewashing the timber-work both ignition and the spread of a fire may be retarded. The third method of construction is in use where large plants, more or less permanent in their nature, are erected. Examples have been cited above. The fourth method is quite common in coal-mine surface-plants in Westphalia, Germany, but is not often met with in the Western States of America. It merits consideration by the mining engineers of the United States. The use of reinforced concrete is uncommon, but some examples are to be recorded in Nevada. The Midway mill, erected in the early days of Tonopah, was constructed with steel frame and corrugated-iron siding, and contained a reinforced-concrete ore-bin. In the Belmont mill and the Goldfield Consolidated mill, reinforced concrete has been used. In Nevada, brick construction, for obvious reasons, is not used for mine-plants, but in the case of electrical installations for power-plant and sub-stations this method of construction is common.

The critical parts of a mine surface-plant are: the blacksmith-shop, boiler- and power-plant, change-quarters, woodshop, oil-storage, rope-house, and the structures about the shaft-mouth. With order, cleanliness, and a proper segregation of buildings, there is little chance for a fire, and yet carelessness on the part of any individual may be the cause of starting a fire, and consequently protection must be afforded by fire-plugs and hose-reels attached to the water-system. Usually the mine-plant is compactly arranged, and several plugs and hose-reels may serve all purposes. Small fire-extinguishers in each building are also considered necessary.

Underground mine-fires are serious, as in almost every case there is danger that fatalities may result, and the difficulty and danger of fighting such fires is great. In heavily-timbered mines where there is little or no water, the possibility of a small fire spreading throughout a mine is always present. Fortunately, dry mines usually do not have large areas of heavy ground, and heavily-timbered mines are often wet mines.

The causes of fires in metalliferous mines are:

The presence of combustible materials, such as timber, oils, waste.

Carelessness with candles, lamps, and smoking.

Blasting; remnants of smoldering fuse.

Overheated bearings in machinery.

Short-circuiting and overheating of electric wires.

Spontaneous combustion.

Rapid progress has been made in the development of methods of mining "without timber," and many methods involving a small proportion of timber, as compared with that required by the "square set" system, are in use. Even though we may largely eliminate timbered stopes or, by filling, practically eliminate the fire-risk from them, we still have the timbered drifts, stations, shafts, pump-stations, and winzes. We could almost completely eliminate the fire-risk from these by the use of steel and masonry, but this is practicable only in a few cases, and the mines of the West will undoubtedly utilize timber for many years to come. Underground, a relatively large amount of combustible material must be contended with. The critical places are the shaft-mouth, the shaft, the stations, the drifts, and the winzes. The critical condition is where the timber is dry and comparatively little water occurs. Where water is encountered the fire-risk is lessened. Systematic elimination of combustible material should be the first thought of the engineer. The proper protection from fire where combustible material must be used is the next consideration.

Carelessness with candles may be eliminated by the introduction of electric lighting, and this method of lighting is common in Western mines of any size. Where candles are used, proper receptacles at stations and in stopes should be provided and their use insisted upon. The use of these will go a long way towards eliminating this cause. The setting of lighted candles upon timbers should be prohibited, and the removal of all candles from working-places when miners are leaving should be required. Lamps should be sparingly used, and where possible these should be filled and trimmed outside of the mine. Oil, and particularly oil required in illumination, should not be stored in the mine. Where oil-lamps are used by the miners, some form of solid illuminant should be used. Smoking can be controlled by mine-regulations, and in a heavily timbered mine, or in and about a wooden surface-plant, it should be prohibited. Where possible, the development of a stable, steady-working crew of men is an important factor in preventing accidents of any kind. Good foremanship is essential to this end.

Blasting as a cause of mine-fires is, no doubt, of minor importance. I know of no authentic case where the flame of a blast has been responsible for setting timbers on fire. In the case of the Homestake fire a piece of smoldering fuse has been given as the cause. However this may be, it is necessary carefully to inspect timbered stopes after blasting. A fire originating from any cause during the interim between blasting and the arrival of the next shift may thus be discovered before it has gained any considerable headway.

Overheated bearings of machinery are an infrequent cause of fire. Underground ventilating- and pumping-machinery are the only forms of machinery which would be likely to cause trouble of this nature. Ring oil-bearings on motors and fans have, to a considerable extent, removed this source of danger, but frequent inspection of machinery should be made. Where fans are in use an inspection should be made at least twice a shift by the shift-boss. Ring oil-bearings should be frequently examined and kept filled, and at intervals the oil should be completely removed, and the bearings thoroughly cleaned. Machinery oil should be kept underground in quantities only sufficient for several shifts' use. Machinery, where possible, should be placed in untimbered chambers or, where support of some kind is necessary, masonry, steel, or some fire-proof material should be used. At Virginia City the pump-rooms are heavily timbered and in themselves contain sufficient timber to sustain a considerable fire. The practice is to whitewash these timbers, and to keep such rooms thoroughly clean. While this reduces the fire-risk, still the presence of so much combustible material, and some of it of an oil-soaked nature, must be considered as a risk, and precautions should be taken to minimize it. (See note 1.) Such chambers can be readily provided with hose-plugs and reels. Where machinery of any size is in operation, the presence of attendants is usual, and is an additional safeguard. Oily waste and waste of any kind used in and about underground machinery should have metal containers provided.

Electric wires and apparatus may have been the cause of some fires, but where their installation has been carefully looked after, and they are in the hands of experienced men, it is seldom that a fire can be directly traced to their use. (See note 1.) The proper making of connections, the use of fuses, and automatic circuit-breakers on all apparatus will prevent excessive loads coming upon lines, electric motors, and transformers. At Virginia City transformers are used at several places underground. These are placed in timbered stations, but this practice is open to question. Transformers should be placed in chambers free from timbers, and only material of a fire-proof nature should be permitted in the vicinity. In surface electrical work transformers are placed in fire-proof buildings, and out of contact with wood-work. Where transformers are in use underground, and in the vicinity of combustible material, buckets of sand should be placed where they can be used in extinguishing a fire.[1]

Spontaneous combustion is an infrequent occurrence, for the reason that underground conditions are seldom of a nature that would lead to this cause. Oily waste might, under extreme conditions, develop sufficient heat to ignite combustible material in its neighborhood, but it is seldom that this is present. Fires in mines containing heavily sulphureted ores are claimed by some writers to be due to the heat produced by great pressure. While this cause is not of great importance, a careful mine-manager will not overlook it.

II. FIRE-PREVENTION.

The first line of defense in the prevention of fire is a proper set of mine-regulations covering the use of candles, lamps, oils, and other combustibles used underground; the second is the enforcement of these regulations. Without discipline mine-regulations are of little avail. The third line of defense is the practice of a fire-drill at frequent intervals. This practice should include drill in the use of fire-fighting apparatus, the training of a suitable fire-fighting squad, and the accustoming of the men to answer an emergency fire-call, so that a possible panic among the men on the alarm of fire may be avoided. The drill should be segregated into the drill of the fire-fighting squad at least once a month, and the drill of the whole mine-force in answering a fire-call at least once in three months. The fourth line of defense is the use of watchmen, whose business it should be to inspect the mine-workings after a shift has departed or at frequent intervals when the mine is shut down. These men serve to check up careless miners and prevent a small fire from spreading.

Facilities for fighting incipient fires should be provided, and placed at readily accessible points in the vicinity of the places where there is any considerable amount of combustible material. These places would be at shaft-stations, the sill-floors of timbered stopes, timbered winzes, and shaft-collars. Fire-extinguishers of a simple type would be the means provided. Water-pipe line should be laid to large timbered stopes, and hose-connections provided. Hose-reels should be placed at the critical points. The whole system should be standardized so that hose, couplings, etc., could be transferred, and used in any part of the mine. At the surface a reserve supply of hose should be provided. A monthly inspection of fire-fighting appliances should be provided for.

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[1] The Factor of Safety in Mine Electrical Installations, *Technical Paper No.* 19, *U. S. Bureau of Mines* (1912).

Steel fire-doors, or wooden doors protected by tin sheets, set in concrete bulkheads, should be provided in cross-cuts and drifts at such points as would enable the spread of a fire starting in a stope to be checked. These would only be used in the event of a considerable fire. Such doors must be constructed so as to admit of opening from either side. In important tunnels of any great length and where timber is used several fire-doors might well be placed so as to divide the length into sections. In lieu of fire-doors a length of 100 ft. of the tunnel could be supported by masonry instead of timber, and this would localize a fire. In tunnel-fires fire-doors cannot always be reached, and the conditions may be such that temporary bulkheads cannot be constructed. In such cases the masonry zone might prevent the fire from spreading through the entire length of the tunnel. In railroad tunnel-work, where, from financial conditions, timber must be used instead of masonry, masonry zones might well be used. The recent fire in the Chilcoot tunnel of the Western Pacific railroad is a case in point. A main working-shaft in a mine would admit of fire-localization in the same way. In place of masonry, steel tunnel-sets or shaft-sets could be used. While perhaps general rules for the placing of fire-stops and doors could be formulated, the conditions at each mine would have to be studied and the locations of these safeguards determined in such manner as to best meet those conditions.

O. F. Heizer has informed me that at the Seven Troughs Coalition mine, Seven Troughs, Nev., hinged wooden doors, protected by $\frac{3}{16}$-in. steel sheets, are placed at the second set below the collar of the incline shaft (75° incline). These doors are held by trickers controlled by a lever. Slots are cut so that interference with the hoisting-cable is avoided. Ten sacks of sand are stored close by the shaft. In the event of a fire on the surface the surface-men have instructions to throw the doors and place the sand over them. He informs me that this device worked very well on one occasion when a fire took place in the surface-plant. He also informs me that at the Nevada Hills mine, Fairview, two steel sheets are arranged at the mouth of the shaft so that they may be slid over the opening of the compartment by throwing a lever. No special rollers are used, the sheets simply sliding on angle-irons.

To one familiar with conditions in Western mines, it is a matter of surprise that advantage has not been taken of the automatic sprinkler, which is in use for the protection of warehouses, factories, public buildings, and the like. I know of no instances where this system is used underground, and only in a few instances, notably in Montana, have they been used in metallurgical plants. The system as applied to buildings consists of a network of pipe-lines supplied by a pressure-tank. The pipes are so distributed as to admit of a sprinkler-head being placed in the center of each 10-ft. square. The sprinkler-heads are placed close to the ceiling. Each head is provided with an opening which is closed by a non-corrosive button. The button is held in place by a two-piece metal strip. The pieces of the strip are held in place by a soft solder which has a melting-point sufficiently low to be speedily reached by any fire in the vicinity. With the melting of the solder the pieces fall apart, the water-pressure forces the button out and the water spurts out against a spreader which throws the spray over a considerable area. A melting-point of 155° F. is used in the well-known Grinnell automatic sprinkler. The complete installation of an automatic sprinkler-system in a mine, while not necessarily impracticable, would in most cases be unnecessary. As has been mentioned, the critical places are the shafts and shaft-stations where the timbers are dry. Shaft and stations might well be protected by such a system. Timbered pump- or machinery-stations, on account of the value of their contents and their importance to the mine, could also be protected in this way.

The installation of an automatic sprinkler system in a shaft is represented in Fig. 1. Two main points in the design of

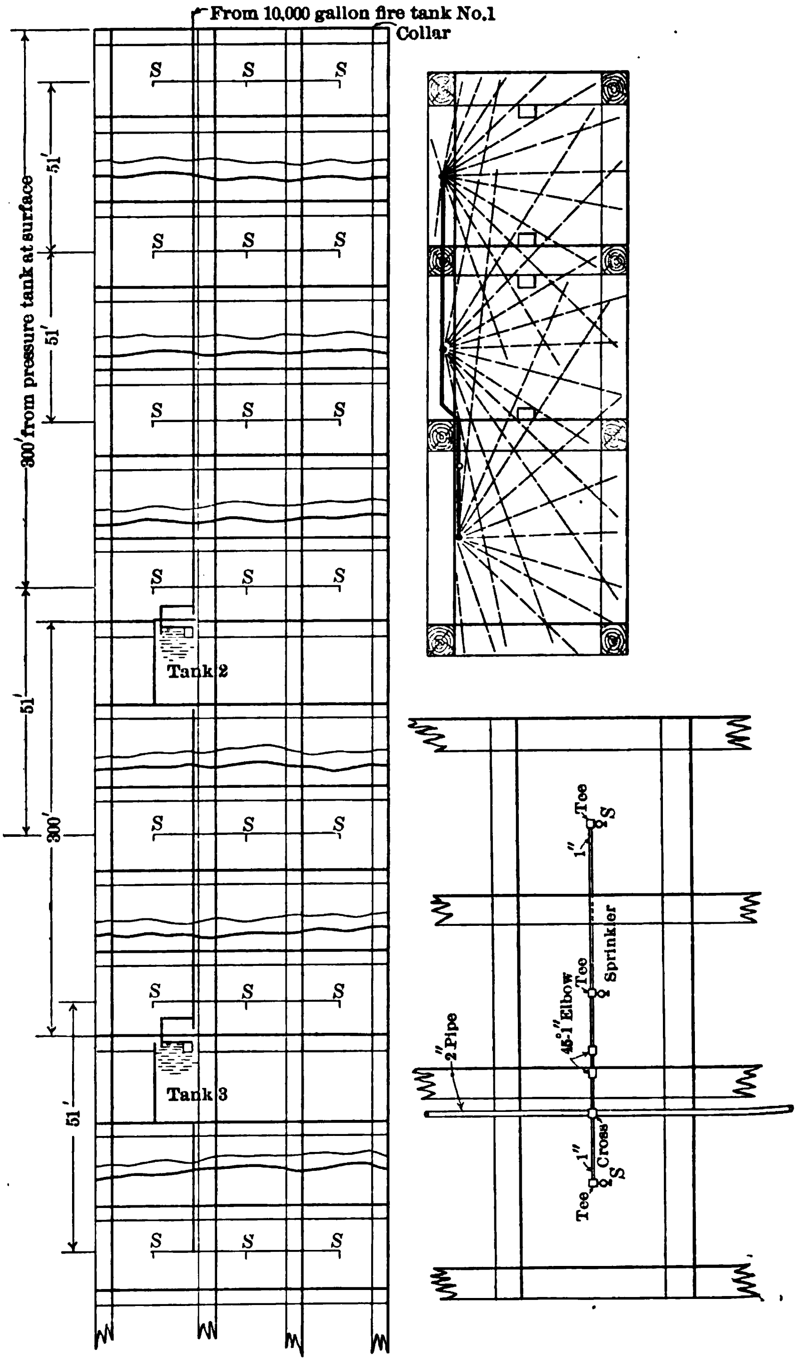

Fig. 1.—Automatic Sprinkler Installation for a Mine-Shaft.

such a system need consideration: one is the question of water-pressure; the other is the use of protective coating upon the pipe so that scale and rust cannot form and clog the sprinkler-heads. In a deep shaft it is doubtful whether a continuous pipe-line could be employed without the use of extra-heavy pipe and fittings. By placing pressure-tanks at intervals of from 300 to 500 ft., ordinary or "extra-strong" pipe and fittings could be used. The intermediate tanks could be supplied from the pipe-line above by means of valves operated by float. The use of galvanized pipe and fittings would eliminate ordinary troubles of corrosion.

In each compartment of the shaft, at intervals, sprinkler-heads, point downward, would be placed. In order to be out of the way these would be placed back against the lagging, but this position would not prevent their effective operation. A three-compartment shaft would be protected by making the intervals between sets of sprinklers 50 feet.

The cost of such a system is nominal compared to the amount of damage a shaft-fire might cause. I have estimated the cost of pipes, sprinklers, and pressure-tanks (every 300 ft.) for a shaft 1,000 ft. deep to approximate $500, or $0.50 per foot. In Fig. 1 a 2-in. pipe is assumed as sufficient. In the cost of the system should be included a fire-tank on the surface. Shaft-stations would be protected by extensions of the pipe-system along the center-line of the station and the spacing of sprinklers every 10 feet.

The advantage of such a system is its automatic operation. A fire arising in an obscure portion of the shaft would be taken care of and extinguished before it could spread. The shaft, the main exit of a mine, could thus be always protected and maintained in a workable condition. The possibility of saving life in the case of a fire would be greatly enhanced if shafts could without question be kept open.

By extending the sprinkler-system along each drift for 50 or 100 ft. a fire could be prevented from spreading to the shaft.

An automatic system to be satisfactory should be inspected at frequent intervals and such a system, carefully installed, would not require an overall inspection more than once in three months.

A telephone-system is an indispensable part of the equipment of a large mine. Not only is it necessary to install telephones at stations, but important winzes and stopes also should be provided with this appliance. The prompt warning of the men may be the means of preventing loss of life. This could be done by telephone or by the use of flash-signals where incandescent lamps are used. In Virginia City an "all out of the mine" signal is in use, and the electric lamps by a given number of flashes convey this signal.

Fire-fighting helmets are now considered necessary at all large mines, and the training of men in their use should be a part of the fire-drill. Oxygen-helmets also should be provided for possible use in rescue-work as well as for the attack of smoldering fires which would produce such an amount of smoke as to prevent near approach. So well has the subject of the construction and use of the oxygen-helmet been discussed that further comment is unnecessary here.[2]

All underground air-lines should be provided with connections at stations, winzes, and stopes. These connections should be maintained in working-order at all times. In the Belmont fire a connection on the air-line at the 1100 station would have been the means of saving life. Where a water-pipe line is not in use in a working-shaft, connections should be made at the surface, so that the air-line could be used for purposes of bringing water to a fire. It is preferable to have two lines of pipe in a shaft, one for water and one for compressed air.

III. FIGHTING MINE-FIRES.

The first essential in fighting a mine-fire is to get all of the men out of the mine. When this has been accomplished, a plan of action can be decided upon and carried out under competent direction. The usual steps may be stated as: bulkheading, with the object of localizing the fire; laying of hose and the bringing of a water-supply to the scene of the fire.

The fire-fighting squad necessarily works upon the incoming-air side of a fire unless equipped with smoke- or oxygen-helmets. By carrying air-lines along with hose-lines, fire-fighting squads have been enabled to approach close enough to a fire from the "lee side" to do effective work. It requires courage and daring on the part of the fire-squad to perform work of this kind, but instances are not uncommon where stubborn fights of this kind have been made.

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[2] **The Use and Care of Mine-Rescue Breathing Apparatus,** ***Miners' Circular No. 4, U. S. Bureau of Mines*** **(1911).**

Where it is impossible to approach the fire close enough to fight it with water, two methods may be used: one is the bulkheading of the fire on both sides and the closing of all winzes leading from the fire-zone; the other is to seal the mine and fill the workings with a gas which will prevent combustion. In the former method the fire is left to smother out, and this may take considerable time. The availability of the oxygen-helmet renders it possible to construct bulkheads where it would have been practically impossible without the use of this appliance. In fighting a fire by the second method, steam, sulphur dioxide, and carbon dioxide have been proposed as gases suitable for the purpose. Steam is the agent most used. If a supply of sulphur could be speedily obtained, it might be possible to use this reagent in temporary burners arranged so as to discharge the gases into the intake air-ways. Snelling[3] has given details of the method. Carbon dioxide is difficult to generate in sufficient quantity, and its use is almost out of the question save as it is generated by the fire itself. Where it is impossible to bulkhead a fire, and steam or other agent is out of the question, the flooding of the mine is the next expedient. If it is impracticable to flood the mine, the turning of water down the shafts after sealing all openings to the mine is then in order.

The great danger in all mine-fires is the rapid filling up of the workings with smoke and poisonous gases (CO). To one who is familiar with fires this is the most striking thing. The comparatively restricted workings of a mine fill up in a very short time on account of the air-currents, which, while normally moving sluggishly, under the increased temperature rapidly acquire velocity. The presence of fire-doors at intervals may be the means of preventing workings from being completely flooded with irrespirable gases. If it were possible to stop all air-currents in a mine by doors suitably placed, the fire and the gas could be prevented from spreading outside of a restricted zone.

As in the case of surface-fires, "being prepared" is the key to the situation. Mine-superintendents should carefully consider the possibilities of a fire, and make every preparation beforehand, even to laying out a method of procedure for fires occurring in different parts of the mine. The drafting of a plan of action, the provision of apparatus and means for carrying out this plan, as well as the drilling and training of the men who are to carry the plan into execution, will go a long way towards preventing confusion, delay, and loss of control at the fire-signal. The provision for marking the passages leading to shafts and exits should be carried out in all large mines. A careful study of the ventilating-currents in a mine and a consideration of the effect of a fire in changing their direction also form necessary parts of any fire-fighting plan. Ventilating-plans of the mine under varying conditions should be prepared, and their study made a part of the drill by the fire-fighting squad. The effect of turning a stream of water down a shaft upon the air-currents deserves mention. In most cases a stream of water turned down an upcast shaft has the effect of reversing the air-current. In the Giroux fire the upcast shaft which was on fire was provided with a water-pipe, pierced with holes, close to the collar of the shaft. An attendant at the surface (without authorization) turned the water into this pipe, and thus reversed the direction of the air, causing the shaft to act as a down-cast. The Alpha shaft, which was the down-cast, became an upcast, and the miners escaping by this shaft were killed by the gases.

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[3] *Trans.*, xxxix., 550 (1908).

IV. LEGAL REGULATIONS.

An examination of the mining-laws of the Western States indicates no general tendency to comprehensively cover the subject of mine-fires. As an example, the regulations provided by the State Mine Inspector Law of Nevada may be cited. These regulations are grouped as follows:

Regulations Relating to Egress from Mines.

"Section 19.—All shafts shall be equipped with ladders, and shafts more than 200 feet in depth, inclined more than 45 degrees from the horizontal, equipped with hoisting machinery, shall be divided into at least two compartments; one compartment to be divided off and set aside for a ladderway. The ladders shall be sufficiently strong for the purpose demanded, and landings shall be constructed not more than 30 feet apart; said landings to be closely covered, except an opening large enough to permit the passage of a man. A landing shall be constructed in manway at all working levels."

"Section 20.—In every mine within this State, if more than 200 feet in depth, where a single shaft affords the only means of egress to persons employed under-

ground, and the ladderway compartment is covered by a non-fireproof building, it shall be the duty of the operator of said mine to cause said ladderway to be securely bulkheaded, or a trap door placed over same at a point at least 25 feet below the collar of the shaft, and if a trap door is used, it must be kept closed, or so arranged that it can be closed from a point outside of the building by the releasing of a rope, and below this bulkhead or trap door, if the shaft is situated on a side hill, a drift shall be driven to the surface, and if the shaft containing said ladderway may be otherwise situated, this drift shall be driven on the level to a safe distance, but in no case less than 30 feet beyond the walls of the building covering the main shaft, and from such a point a raise shall be made to the surface. The said raise shall be equipped with a ladderway, and it, together with the drift connecting with the main shaft, shall be kept in good repair, and shall afford an easy exit in the event of fire."

"Section 21.—Whenever the exit or outlet from a mine is not in a direct or continuous course, signboards, plainly marked, showing the direction to be taken, must be placed at each departure from the continuous course."

"Section 28.—It shall be the duty of every operator to provide every tunnel or adit level, the mouth of which is covered by a house or building of any kind, with a door near the mouth of the same, that can be closed from the outside of the building by a pull wire or cable in the event of fire; inside of door a raise shall be run to connect with surface, thus affording a means of exit in the case of fire."

Regulations Relating to Structures over Shaft-Mouths.

"Section 27.—It shall be unlawful for the operator of any mine within the State to erect any structure over the shaft of any mine, except head-frames necessary for hoisting from said shaft or outlet, and the hatch or door necessary for closing such shaft or outlet; *provided, however*, it shall be lawful to erect a house of non-inflammable and fire-proof material over such shaft or adit to protect the men working at such point. In the case of existing houses covering mouths of shafts or adits, it shall be the duty of the superintendent of the mine to cause the immediate removal of all inflammable material stored therein, and it shall be the further duty of such superintendent to prohibit the storage of any inflammable material 30 feet from the exterior walls of any housing hereinafter built."

Regulations Relating to Inflammable Materials Used Underground and on Surface.

"Section 16.—All timber removed shall, as soon as practicable, be taken from the mine, and shall not be piled up and permitted to decay underground."

"Section 22.—Use of gasoline underground is forbidden."

Regulations Relating to Fire-Fighting and Control of Mine-Fires.

"Section 41.—At every mine in this State, employing forty or more men underground, there shall be kept on hand at all times, in good working condition, at least two smoke helmets of a design to be approved by the State Mining Inspector, and which helmets shall at all times be subject to his inspection. For every additional fifty men so employed an additional smoke helmet shall be provided."

These regulations are, at best, only fragmentary, but they indicate an effort to provide certain things that a mine-operator must do. Most mine-operators are willing to conform to any reasonable regulations, but until a more or less complete code is drawn up and incorporated in our State mining-laws, we must expect to find them somewhat backward in taking the initiative, except in those cases where either bitter experience or broad training has stimulated operators to take the subject up in detail.

V. Proposed Regulations.

With the object of inviting discussion, and of giving point to the generalizations in the foregoing, I have written the following regulations. The difficulty of providing for every emergency that may arise in a mine-fire, and of meeting the miscellaneous conditions that are present in Western metal-mines, is apparent, and in preparing these regulations I have endeavored to maintain a conservative rather than an extreme position.

1. *Egress from Mines.*

For mines deeper than 200 ft., and employing ten or more men underground, operators must observe the following:

1. Two outlets must be maintained in good condition at all times during the operation of the mine.
2. Where two or more shafts are in use, two of such shafts must be provided with ladder-ways in separate compartments, and such ladder-ways maintained in good condition during the operation of the mine.
3. Where ladder-way shafts are inclined at a greater angle than 45°, landings closely boarded, and with openings just sufficient to allow the passage of a man, must be provided at intervals not greater than 30 feet.
4. In mines operated through three or more working-shafts, where each shaft is provided with hoisting-machinery, a ladder-way may be provided in but one shaft.
5. Between levels at least one winze must be provided with a ladder-way, and the same maintained in proper condition where the level is in use for extraction of ore, or ventilation.
6. On each level the direction to shafts and winzes, used as exits in cases of emergency, must be clearly indicated by sign-boards.

2. *Mine Surface-Structures.*

1. None but fire-proof buildings are to be permitted over shaft or tunnel-mouth.

2. Where shaft-mouths are open, wooden head-frames and bins may be permitted.

3. Non-fire-proof buildings must not be placed nearer than 50 ft. to any shaft-mouth or tunnel-entrance.

4. Mine-plant buildings of non-fire-proof construction must be separated from each other by a space not less than 30 ft. wide.

5. In a mine surface-plant where non-fire-proof construction is employed, hoisting-machinery, boiler-plant, blacksmith- and machine-shop, timber-framing shop, change-house, and storage of supplies must be placed in separate buildings separated by a fire-space not less than 30 ft. wide.

3. *Inflammable Materials on Surface and Underground.*

1. Where fire-proof shaft- and tunnel-houses are in use, timber or other inflammable material in excess of sufficient for one shift's use must not be stored in such buildings.

2. Timber-yard or timber-storage sheds must be placed not less than 75 ft. from any mine-building.

3. Lubricating-oils and inflammable fluids must be stored in fire-proof buildings separated by a distance of not less than 75 ft. from other mine-buildings.

4. Metal containers must be provided for the storage of all waste used in wiping and cleaning machinery. They are to be provided for each machinery-room in the surface-plant, and for each group of machines underground.

5. Lubricating- and illuminating-oils in excess of a 24-hr. supply must not be stored underground or in surface-buildings other than the oil-storage building.

6. Gasoline and substances of like nature are not to be used underground.

7. Timber in excess of a 24-hr. supply is not to be stored underground.

8. Powder-drifts which are used for the temporary storage and handling of powder underground, must be cleared of all paper, empty boxes and rubbish, at least once each 24 hours.

9. Where timber is framed at one or more points in a mine, these places must be cleared of chips and rubbish, at least once each 24 hours.

4. *Surface and Underground Regulations.*

1. Where mine surface-structures are of non-fire-proof construction a sufficient water-supply must be provided for fire-fighting purposes. One fire-plug and hose-reel must be placed not less than 25 ft. from the shaft-mouth, and such other fire-plugs placed as, in the judgment of the State mine-inspector and mine-superintendent, shall be sufficient to quench any ordinary fire.

2. At shaft-house, in timber-framing building, power-plant, change-room, and such other parts of the surface-plant as the mine-inspector may decide, at least two fire-extinguishers of an approved type must be placed and maintained in proper working-order. They must be placed in a conspicuous and convenient place.

3. Smoking shall be prohibited in non-fire-proof surface-plants.

4. Where more than a nominal number of lamps using illuminating-oils are in use a separate lamp-house shall be provided and all filling, cleaning, and trimming done in this building.

5. Where water-pipes are not installed in shaft, connections must be provided with the surface water-system so that the compressed-air pipes may be used to bring water underground in the case of a fire.

6. Connections must be made with air-pipes at each station, whether the same is in active use or not. These connections must be such that air can be turned into the station.

7. Where candle illumination is used underground metal sconces must be provided at all timbered stations and stopes and miners required to use same.

8. The placing of lighted candles on timbers without proper protection is prohibited.

9. Smoking is prohibited in timbered mines.

10. At all timbered stations which are in active use and where water-supply and hose-lines are not provided, at least two fire-extinguishers of approved type shall be placed and maintained in working condition.

11. Where electric illumination or power is used underground or in surface-plant, line-installation and protection must be in accordance with the Electric Code of the National Board of Fire Underwriters.

12. Transformers, where used underground, must be placed in fire-proof chambers and, at each bank of transformers, sand-boxes must be placed and a supply of sand maintained.

13. In heavily-timbered stopes fire-inspection must be provided after each shift leaves and before the new shift comes on.

14. Air-tight doors shall be constructed in each level, where practicable, between upcast and down-cast shafts. Such doors should be preferably of fire-proof construction.

5. *Regulations for Fire-Fighting and Control of Mine-Fires.*

1. The mine-foreman, under the direction of the mine-superintendent, shall have charge of all fire-fighting operations.

2. The mine-foreman shall designate certain assistants to constitute a fire-fighting squad, and assign duties to such assistants. The names of such assistants shall be posted at shaft-mouths as the fire-fighting squad.

3. The mine-foreman shall conduct a general fire-drill at least once each three months and at that time examine, test, and report to the superintendent the condition of all fire-fighting apparatus, exits, and ladder-ways. This report shall be in writing, and shall receive the signature of the State mine-inspector on his next inspection-visit after the fire-drill.

4. In mines employing 50 men, at least two smoke-helmets of an approved type shall be maintained in proper working-condition at all times. For each 30 men in addition an additional smoke-helmet shall be installed and maintained in working-condition. The foreman shall instruct the fire-fighting squad in their use.

5. In mines employing 200 or more men underground, at least two oxygen-helmets shall be maintained in proper working-condition. The foreman shall designate certain men for instruction and practice in the use of such helmets.

5a. In mines employing 200 or more men underground, a "pulmoter" in proper working-condition must be provided. The foreman shall instruct the fire-fighting squad in its use.

6. A general signal to indicate a surface-fire and one for an underground fire shall be designated.

7. Where practicable, an "all out of the mine" signal shall be designated and used in emergency. This signal must be used only by persons to be designated by the superintendent.

8. Detailed instructions to surface and underground men as to what to do in the case of fire are to be posted at shaft-mouth and underground stations.

VI. Conclusion.

This paper is written in the hope that some discussion will result. The use of fire-proof construction, the comparative cost of such construction, methods of preventing the spread of, and of fighting, mine-fires, and mine-fire regulations are subjects upon which many of the members of the Institute can no doubt contribute much of importance.

[TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS.]

Recent Developments in Open-Hearth Steel-Practice.

BY N. E. MACCALLUM, PHŒNIXVILLE, PA.

(Cleveland Meeting, October, 1912.)

ALMOST half a century has passed since the Siemens brothers, after tedious and costly experiments, finally began the manufacture of open-hearth steel. The furnace of that time was very small, having a hearth-capacity of from 3 to 4 tons, and we can appreciate the wonderful development that has taken place in those 50 years by the knowledge that to-day furnaces of 50 times the original capacity are in operation. As the new steel process assumed importance, and furnaces began to grow in number and in size, a question arose, to which no satisfactory answer has ever been given, and that was: "What is the safe and economical limit of the open-hearth furnace?" Various stopping-places have been suggested, and 20, 30, 40, and 50 tons have each been advocated, but all such arbitrary restrictions have been passed, and 60 tons would probably represent the average furnace, although in a few instances from 70 to 80 tons are regularly tapped into one ladle. The furnaces which are the subject of the present paper have gone far beyond even these limits, and since, in my opinion, they mark another step in the progress of open-hearth steel, they will be considered fairly and frankly, in order that those to whom the matter may be of interest may be able to judge and form their own conclusions.

Short accounts of the work of these furnaces have appeared from time to time in various technical journals, of which the following is a brief *résumé*.

These furnaces are the property of the Phœnix Iron Co., of Phœnixville, Pa., and their distinctive and important feature is, that they charge to twice the capacity of one ladle, and tap simultaneously into two ladles, with a device for controlling the flow into either. In this way the capacity of the furnace is not limited, as formerly, but heats of any size within the capacity of two ladles may be charged.

The ultimate reason for charging furnaces beyond the normal amount is not that large heats may be made, but because by making large heats a material increase of product is obtained.

Fig. 1 is a view of a furnace with divided spout, tapping into two ladles. A plan of the divided spout, pits, and ladles is given in Fig. 2. In order to make the branches of the spout as short as possible, one ladle is placed slightly in advance of the other, which also enables the trunnions to clear. Fig. 3 shows a plan and a front-elevation and Fig. 4 a side-elevation of the device for controlling the flow of steel into either ladle. Its purpose is not to stop, but merely to check, the flow. The levers by which it is operated are placed from 20 to 30 ft. from the center of the furnace, and the controlling-rod is protected by fire-brick, much the same as the ladle-rod.

The first of these furnaces, here designated as A, B, and C, was completed early in 1909. The area of the hearth, which had been decided upon after a few friendly visits to neighboring plants, was 32 by 14 ft., and was a fair average of furnaces charging from 50 to 55 tons of materials. At that time no thought had been entertained of breaking away from the established practice, but, on comparing the capacity of this hearth with those of our smaller furnaces, it proved to be so much in excess that the question was raised as to the advantage of a large hearth if it could not be fully utilized. Various experiments were then undertaken to determine the feasibility of using two ladles, and the best means of controlling the flow into either ladle, which resulted in our present practice.

Furnace B, completed in September, 1909, was the same in all respects as Furnace A, except that the hearth was lengthened 4 ft., making it 36 by 14 ft. in area.

Furnace C was built in 1911, the size of the hearth being increased to 42 by 15 ft. The size of the regenerator-chambers was also increased. These chambers were designed to meet existing conditions. The air-chambers of Furnaces A and B are 9 ft. wide and the gas-chambers 6 ft. wide. For Furnace C the air-chambers are 12 ft. wide and the gas-chambers 8 ft. wide. The length of chambers and depth of checker work are uniform for all the furnaces, 18 and 7 ft. respectively. If now these furnaces are rated as 115 tons, 130 tons, and 165 tons capacity, which is the usual weight of ingots tapped from each, it will be noted

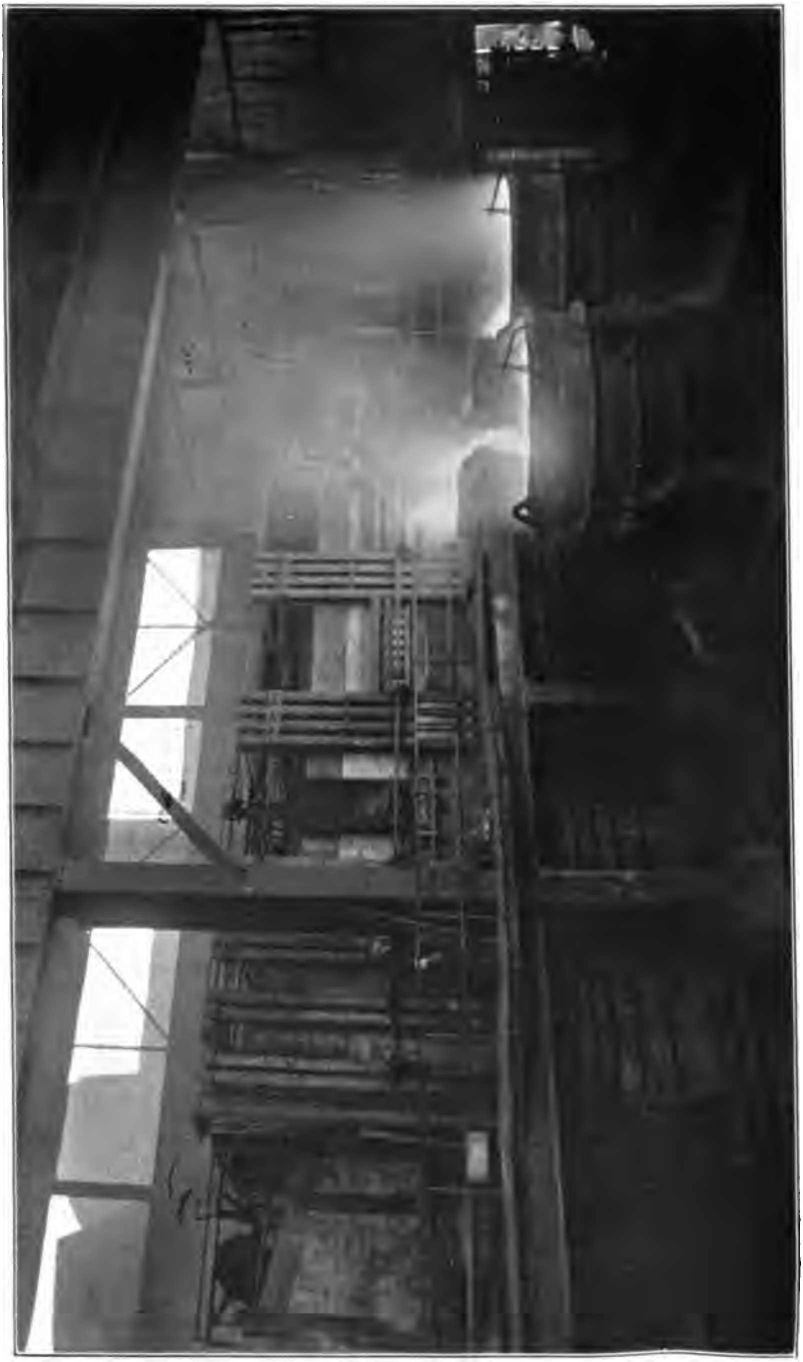

FIG. 1.—VIEW OF OPEN-HEARTH FURNACE WITH DIVIDED SPOUT, TAPPING INTO TWO LADLES.

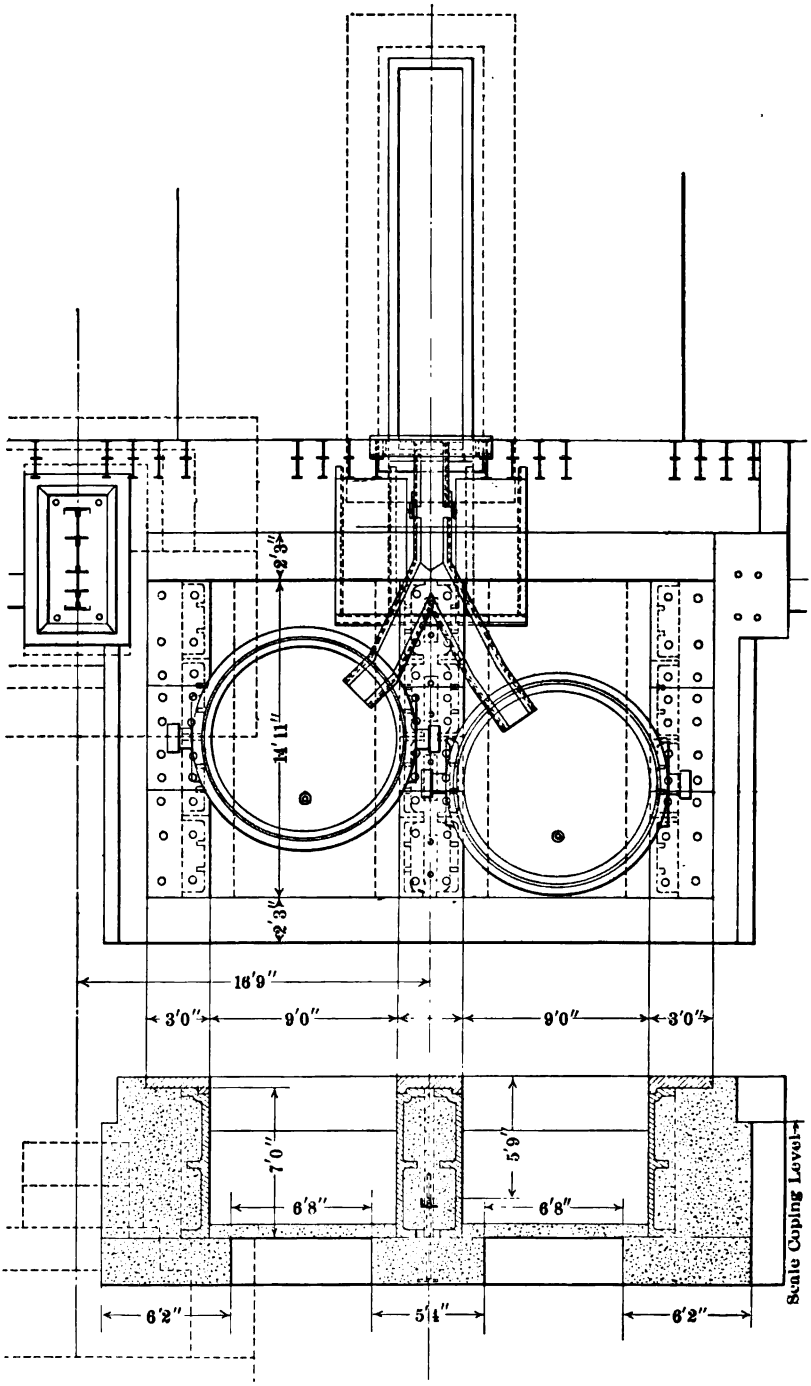

FIG. 2.—GENERAL PLAN OF TWIN LADLE-PIT AND SPOUT FOR OPEN-HEARTH FURNACES.

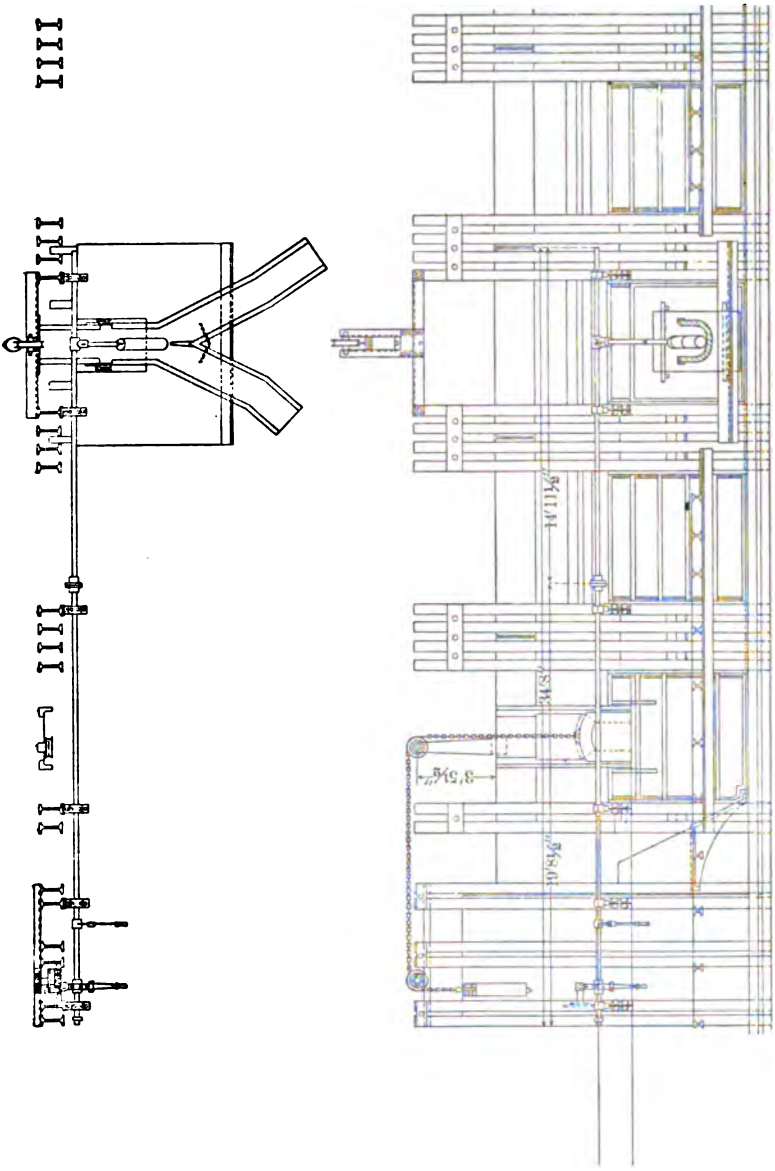

FIG. 3.—PLAN OF STREAM-CONTROLLER FOR TWIN RUNNER-SPOUT AND FRONT-ELEVATION OF OPEN-HEARTH FURNACE.

that the checker-space on each end of the furnace is only equal to about 15 cu. ft. per ton of hearth-capacity; or, one-sixth of the amount generally allowed and recommended by prominent metallurgists. This statement is not made from any desire to criticise or detract from the prevailing views and practice; but it might be worth while to note that, had this practice been followed, the depth of checker-space in Furnaces A, B, and C would have been 46 ft. instead of 7 ft. Or, to state the same thing in another way, the rated capacity of the furnaces, based on checker-space, would be 19 tons for Furnaces A and B, and 25 tons for Furnace C, instead of 115, 130, and 165 tons. It may therefore be safely assumed that any particular merit which

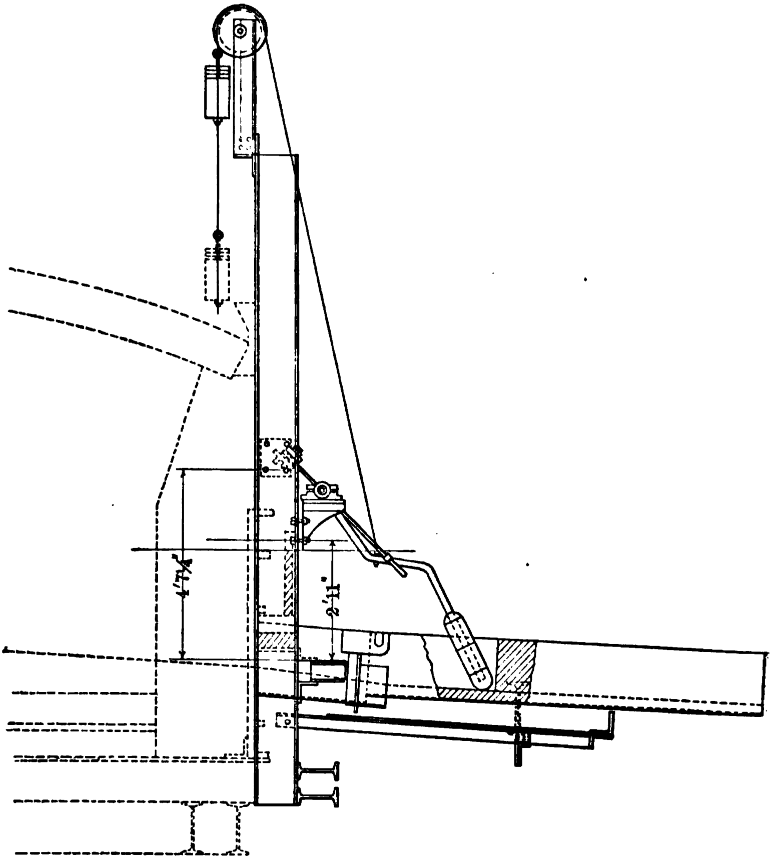

Fig. 4.—Side-Elevation of Stream-Controller.

this practice possesses must be attributed to causes other than the chambers.

The areas of the hearths of Furnaces A and B were practically the same as those of ordinary furnaces rated at from 50 to 55 tons capacity, and that of Furnace C, 42 by 15 ft., giving an area of 630 sq. ft., is almost an exact duplicate of the area of a number of furnaces recently completed and rated at 60 tons. The hearths are of the usual depth and measure from pan-plate to top of fore-plate 5 ft. 0.5 in. The pan is covered with 18 in. of brick and from 10 to 12 in. of magnesite in front of tap-hole. If, therefore, the hearths are of the same size, and the chambers are much smaller than those of many other furnaces, it is apparent that the large tonnage must be due to other reasons, which we shall endeavor to show lie not in the furnace, but in the practice.

On one occasion, in discussing methods of operation with the superintendent of a large steel concern, I asked if he reduced the size of open-hearth heats at the week-end, and the reply was that he did not, as it had been found that a large heat could be made almost as quickly as a small one. That was the open-hearth man's way of broadly stating a truth, which was not intended to be accepted too literally. It is not true, of course, that a large heat can be made as quickly as a small heat, but it is true that steel is made faster in a large unit than in a small one. This seems an obvious fact, yet it is one that receives but little attention and is rarely considered in comparing the work of one furnace with another. In most works, the number of heats which each furnace makes weekly constitutes the basis of comparison; but it is evident that for furnaces of different capacities it would be of little value, since the small furnace might make more heats and yet not equal the tonnage of a larger furnace, or, in furnaces of similar capacities, one might be delayed with repairs to hearth or otherwise and show a decreased tonnage, although it might be a faster-working furnace. For a number of years I have used, as a basis of comparison, the time from beginning the charging until the tap-hole is open, divided by the product, which gives, in minutes per ton, the rate at which steel is being made. This gives a reliable means of determining the speed with which any furnace is capable of making steel, and also its ability to compete with other furnaces. Using this method of comparison, Table I. has been compiled from furnace-records of the Phœnix Iron Co. In every case the record covers a period of not less than six months and should be fairly representative of the work done by each furnace. The smaller furnaces are of the older type with regenerators underneath the hearth, but did efficient work and compared well with other furnaces of similar construction. Furnace C now stands on the site formerly occupied by two of the smaller furnaces, Nos. 5 and 6, and it is interesting to note that Furnace C is making steel just a little more than three times as fast as No. 5 formerly did.

TABLE I.—*Data of Furnace-Practice.*

Furnace.	Hearth-Area.	Weight of Ingots.	Time Per Ton.
		Tons.	Minutes.
No. 3	17.5 by 12 ft.	30	32.79
No. 5	20 ft. 3.5 in. by 12.5 ft.	40	26.48
No. 1	24 ft. by 12.5 ft.	50	19.66
A	32 by 14 ft.	115	11.50
B	36 by 14 ft.	130	10.50
C	42 by 15 ft.	165	8.68

A study of Table I. will reveal two significant facts: One, that the economical limit, so far at least as time is concerned, has not been passed, since Furnace C is shown to be making steel 21 per cent. faster than Furnace B. The other, the striking decrease in minutes per ton as the hearth-area and unit of steel increase, and it is on this fact that the practice of the Phœnix Iron Co. is based. The extent to which any furnace may add to its product will be proportional to the excess charged over the normal one-ladle heat, but from the experience of the Phœnix Iron Co., it may be affirmed that by doubling the normal charge the product of any furnace can be increased by from one-fourth to one-third; that is, if a furnace charging single-ladle heats can produce, say, 600 tons weekly, it will by charging double-ladle heats produce from 750 to 800 tons. This can be readily demonstrated by operating any furnace alternately on single-ladle heats and half-ladle heats, and comparing the number of minutes required in each case to make a ton of steel.

Why should steel be made faster in large units than in small? No particular reason can be assigned, but there are several probable causes which contribute to that result. The first and most important is the economy in time; that is, in a given time more steel will be produced, or the same amount of steel will be produced in less time. This can be better understood by dividing into two periods the time required to make a heat of steel, the first period being the time required to charge and melt, and the second period the time from when the lime floats to the surface of the bath until the heat is ready to be tapped, known as the "working" of the heat. In the first period the double-ladle heat will not show much gain over the single-ladle heat, as it will take almost twice as long to charge and melt, but in the second period, when spar, ore, and other additions are made, the double-ladle heat gains a decided advantage, since it takes but little longer to "work" a large heat than a small one. At various times during the past three years single-ladle heats have occasionally been made on Furnaces A, B, and C, and a careful scrutiny of the records shows that the average time required to make a ton of steel on those heats was 12.2 min., while the average time at which furnaces were running on double-ladle heats was 9.7 min., showing that it required 25.7 per cent. more time to make a ton of steel on a single-ladle heat than on a double-ladle heat.

As a second cause, it is evident that twice as many single-ladle heats will be required to produce the same tonnage as that from double-ladle heats, therefore one-half the time which would be required to make repairs to the hearth in the former case will be available for making steel in the latter. Then again, each heat of steel that is made, whether large or small, entails two extremes of temperature, viz.: the cooling of the hearth and furnace when cold stock is introduced, and the maximum temperature necessary to tap, at which time the hearth and furnace must be raised to at least the same temperature as the steel. It will therefore take twice as many heat-units to raise the furnace and hearth to this temperature to make two single-ladle heats as would be required to make one double-ladle heat, just as in heating water or other substance in a refractory vessel, less time will be required to heat a certain amount in one portion than to divide that amount into two portions and heat each separately, the difference in time being that required to heat the container a second time.

A third cause is radiation. Campbell,[1] in reference to heat-distribution in an open-hearth furnace, says: "Roughly speaking, about one-half of all the heat supplied to an open-hearth furnace is lost by radiation and conduction." That statement concerns the present paper only to the extent that the more frequently a furnace is raised to a tapping-temperature the greater will be the loss by radiation, as the latter increases with each degree of raise of temperature, so that the fewer heats that are made in proportion to the tonnage, the less frequently will the furnace be raised to a tapping-temperature, and the less will be the loss by radiation. This is simply illustrated by noting the time required to heat a volume of water from t° to the boiling-point, then removing a portion and replacing with water at t°, and again raising to the boiling-point, and repeating the operation until the added portions equal the original volume. The added portions collectively will take longer to boil than the original amount, the difference being equal to the excess radiation that has occurred at the higher average temperature.

These views have been advanced with some hesitancy, but they appear to offer a reasonable explanation why steel is made faster in large units than in small ones. To confirm these arguments further, a statement of the actual tonnage which each furnace has made is given below. All the stock is charged cold, and the fuel used is producer-gas.

Furnace.	Period of Time.	Output.	Time Per Ton of Output.	Fastest Time for One Ton of Output.	Largest Heat.
		Gross Tons.	Minutes.	Minutes.	Gross Tons.
A,	First year	41,436	11.45	8.27	130.04
	Second year	39,910	11.54		
B,	First year	43,900	10.38		
	Second year	43,615	10.44	7.49	146.32
C,	47 weeks	47,200	8.68	6.92	177.77

Furnace C has still five weeks in which to complete the year and should run well over 50,000 tons, which is probably the largest tonnage that has ever been produced from any furnace using all cold raw material.

The saving in cost, which will be proportionately increased as the tonnage increases, depends upon labor, fuel, and repairs.

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[1] *Manufacture and Properties of Iron and Steel*, 3d ed., p. 224 (1904).

The number of men necessary to operate a furnace on a double-ladle heat is the same as on a single-ladle heat, and if the product is increased one-third, it is apparent that the labor will be reduced one-fourth, including all furnace-labor except that of the stock-loaders.

The fuel (gas-coal), covering a period of several months, will not vary much whether the furnace is making large or small heats, and, as is the case with the labor, should show a saving of one-fourth. Furnaces A and B are on the same gas-line as the blooming-mill soaking-pits, and, consequently, it is difficult to get accurate data. Furnace C, however, is supplied from an independent line, and the figures used are based on the actual amount of coal received in the cars. On a run of 22 consecutive weeks, including the entire time from commencing the first heat until shut down for repairs, the average amount of coal used per ton of steel was 547 lb. Just whether this represents a saving of one-fourth is uncertain, but we can estimate what the amount of coal would have been had the furnace been operated on single-ladle heats, in which case the product would have been reduced one-fourth and the amount of coal increased one-third. Furnace C averages slightly more than 1,000 tons of product weekly, so the product would have been 750 tons, and the coal increased one-third would have been 729 lb. per ton of steel. That is, instead of 1,000 tons of steel, with a coal-consumption of 547 lb., on double-ladle heats, we should have 750 tons of steel, with coal-consumption of 729 lb., on single-ladle heats.

Bearing in mind the arguments that were advanced in explanation of the increased product of large heats, the use of pig-iron with an average content of 0.70 phosphorus and 0.70 silicon, with considerable quantities of lower-grade iron as cheapeners, together with the miscellaneous scrap that the market affords, 729 lb. of coal cannot be considered excessive for Eastern practice, particularly covering a period of 22 weeks, so that it is probable a saving of one-fourth is not far wide of the mark. With larger regenerators it is likely a still further saving might be effected.

What has been said in regard to labor and fuel should also apply to repairs. That is, in a given period the wear and tear will be about the same, regardless of the size of the heat or of product. In the run of 22 weeks, the total product from Furnace C was 24,298 gross tons, and the repairs that followed, including roof, linings, checkers, cost of brick, bricklaying, and common labor, was a fraction less than 12 cents per ton.

Having discussed the increased product and economies resulting from the two-ladle practice, what are the objections likely to be raised against this method of steel-making? Probably the first would be the risk of handling such large quantities of steel and the possible consequences in case of accident. In reply, it may be said that in modern open-hearth practice, when linings have been increased in thickness, partly to overcome excessive radiation and partly to prevent accidents, break-outs rarely occur, and as double-ladle practice requires only one-half the usual number of heats to produce the same tonnage, it is evident that that risk, whatever it is, has been cut in half. It may be imagined that serious trouble might result from the chilling of a heat in the furnace, as occasionally happens when the roof falls in. I confess that the first instance of this kind was viewed with some apprehension. It occurred on Furnace B, when a charge of 335,000 lb. was melted; the roof from hip to hip fell in. This happened at 8 a.m. on a Saturday, and during the next day the roof and lining were replaced. Gas was turned in at 5.30 a.m. Monday, and the heat was tapped at 10 p.m. Tuesday, yielding 139 gross tons of ingots. A similar experience occurred another time, but in neither case were any difficulties presented other than would be encountered in smaller heats. The divided spout and controller have at all times worked satisfactorily, and no trouble has been experienced in dividing the steel, although occasionally one ladle may receive more than the other, but generally both ladles are evenly filled.

Another objection that might be raised is, the difficulty the mills may have in taking care of the product in large units, but a little consideration will show that when mills are provided with sufficient soaking-pit capacity, it makes but little difference. If the product is supplied from large furnaces, there will be fewer heats, whereas, if supplied from smaller furnaces, there must, necessarily, be more furnaces, and a largely increased number of heats, frequently causing what mill-men call "bunching" of the heats, that is, when several tap together or in close succession. That the large heats are not objectionable to the mill-men is evidenced by the interest with which the tapping hours of the large furnaces are followed.

As to the quality of the steel, it may be said very positively that there is no apparent difference from that made in smaller amounts. The chemical analyses, the physical tests, and fabrication of hundreds of thousands of tons, would surely have detected any difference had such existed.

I have traced, within the limits of this brief paper, the causes which led to the development of the double-ladle practice, and have shown the increased tonnage and decreased cost resulting therefrom, also giving the objections that may be urged against this practice. In conclusion, I may say that my object is, not to present the Phœnixville furnaces as types of fast-working furnaces, as they are not so considered, but to show by the work that they have done what can be accomplished by the practice which they exemplify.

SUBJECT TO REVISION.

[TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS.]

Measurements and Relations of Hardness and Depth of Carbonization in Case-Hardened Steel.

BY MARK A. AMMON, CLEVELAND, OHIO.

(Cleveland Meeting, October, 1912.)

THE two most widely used methods of measuring hardness are the Brinell and the scleroscope. In the Brinell method a hardened steel ball is pressed into the steel under a definite load and the area of the resulting depression is measured. The load in kg. per sq. mm. of the area of the depression is taken as a measure of the hardness. Within certain limits the hardness number so determined is independent of the load or the size of the ball. In the scleroscope method a small weight carrying a diamond point is dropped through a glass tube upon the steel and the amount of rebound is taken as proportional to the hardness.

In both methods the scale of hardness is an arbitrary one; however, they agree in a comparative way, that is, a steel showing 50 per cent. harder Brinell will show approximately 50 per cent. harder scleroscope. Yet upon consideration of the two methods it appears that, upon a steel which has been carbonized, this should not be the case; for the scleroscope method gives almost exclusively the surface-hardness, and the condition of the material below the surface has but a slight bearing upon the hardness number. The results given by the Brinell method should, however, be the exact opposite; for the depth of penetration of a ball pressed into the steel would be dependent to a large extent upon the condition of the material below the surface. This is particularly obvious when it is considered that the size of the ball ordinarily used in the Brinell method is 1 cm. (about ⅜ in.) in diameter under a load of 3,000 kg., while the diamond point of the scleroscope is in the neighborhood of 0.01 in. across, fitted in a hammer of about 0.5 oz. weight and dropping from a height of 10 inches.

It was thought that an investigation into the hardness of a case-hardened steel by means of a combination of these two methods might lead to the development of a process whereby the depth of carbonization could be determined within a reasonable degree of commercial accuracy.

It was seen at once that the ordinary size of steel ball used in the Brinell method would not do, since the size of the depression would be so great that a finished piece would be more or less injured by it. A special attachment was therefore made for the machine, which accommodated a ball 0.25 in. in diameter, and this was used in combination with a weight of 5,000 lb. Instead of measuring the diameter of the resulting depression, the depth of penetration was measured to 0.0001 in.; and by ordinary geometrical methods the following formula was determined to give standard Brinell readings:

$$B = \frac{44,700}{D}$$

B being the standard Brinell reading, and D the depth of penetration in ten-thousandths of an inch.

Duplicate investigations were made upon two different steels, designated as No. 2 and No. 8, and analyzing as follows:

	No. 2. Per Cent.	No. 8. Per Cent.
Carbon	0.233	0.188
Phosphorus	0.011	0.008
Sulphur	0.019	0.020
Manganese	0.53	0.53
Nickel	...	3.30

Each of these steels was carbonized at two different temperatures, 1,650° and 1,750° F. At successive periods, the first of 0.5 hr., the second 1 hr., the third 1.5 hr., and then each hour, one piece of each grade was taken from the furnace and immediately quenched in water, while two others were allowed to cool in the carbonizing-box. The two latter were then reheated and quenched, one receiving a double and the other a single quench. The piece quenched from the box originally, received afterwards a second low heat.

The test-pieces were steel disks 1 in. thick, the two flat surfaces being ground smooth. The diameter of the disks was 1⅛ in.

for grade No. 8, and 2 in. for No. 2. Three of each kind were packed in a cast-iron pot 8 in. in diameter and 12 in. high, making six disks in each pot. They were packed radially and at equal distances from the center. One of each grade had a wire attached to a hole drilled in it, so that it could be pulled out and quenched as soon as the box was drawn from the furnace, the other four being left in to cool with the box.

The time was divided into 10 periods as shown in the tables. Therefore 10 pots each containing the six disks as described above were placed in a very large furnace, giving throughout the heating-chamber a uniform temperature, maintained at 1,650° F. In a second furnace, exactly similar and loaded in the same way, the temperature was maintained at 1,750° F.

After each operation the disks were carefully smoothed with fine emery-cloth; three Brinell and 10 scleroscope readings were taken on each disk, and these were averaged.

In order to prevent confusion of the disks, the following scheme of marking was employed: The figure 2 or 8 was first stamped on them to distinguish the grade; this was followed by the letter H or L to distinguish the disks carbonized at a high (1,750) from a low (1,650) temperature; this was then followed by the letter B, S, or D to distinguish those quenched from the box B, those receiving a single quench S, and those receiving a double quench D after cooling in the box. After these was placed a number from 1 to 10 to represent the order in which the disks were taken from the furnace. For example, 8LS6 meant steel No. 8, carbonized at 1,650°, afterwards receiving a single quench, and the sixth disk taken from the furnace in this series.

One disk of each grade was taken from each box after cooling and a small piece was sawed out of the edge, polished, etched, and the depth of carbonization measured under the microscope, first measuring the depth of carbon greater than the eutectic, then the eutectic, and then that less than the eutectic. This measuring was done by means of a micrometer eye-piece and a mechanical stage, with a magnification of 600 diameters.

The following tabulated data give all of these results, which are reproduced in graphic form in the diagrams.

2LB.

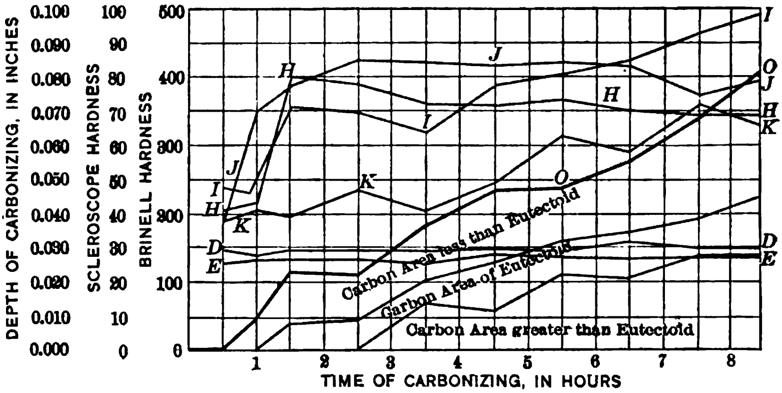
Steel No. 2. Carbonized at 1,650°. 1st Quench from Furnace at 1,650°; 2d Quench at 1,400°.

No.	Original.		1st Quench.		2d Quench.	
	Brin.	Scle.	Brin.	Scle.	Brin.	Scle.
1	186	......	235	41	186	36
2	128	......	229	43	205	70
3	128	......	358	80	196	78
4	135	......	344	78	235	85
5	131	......	319	72	226	66
6	131	......	388	72	246	84
7	128	......	407	74	315	85
8	135	......	425	71	292	84
9	139	......	470	70	372	75
10	122	......	496	68	331	80

2LS.

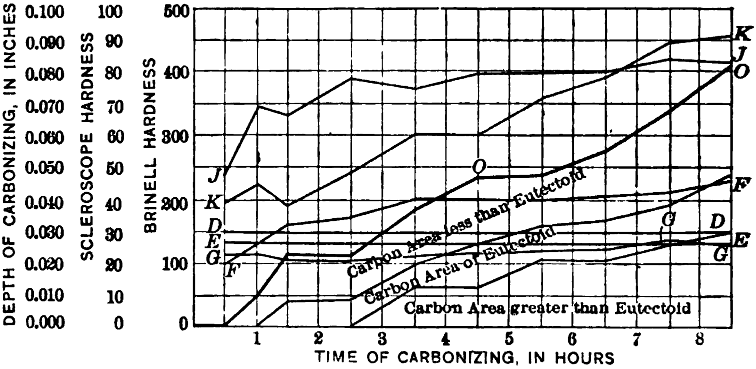
Steel No. 2. Carbonized at 1,650°; Quenched at 1,400°.

No.	Original.		After Carb.		1st Quench.	
	Brin.	Scle.	Brin.	Scle.	Brin.	Scle.
1	126	29	112	20	194	48
2	131	27	112	25	224	70
3	131	29	106	32	190	67
4	131	29	109	35	242	78
5	128	29	114	40	302	75
6	141	30	119	40	302	80
7	137	29	122	40	358	80
8	132	32	124	41	388	80
9	135	30	140	42	447	85
10	129	30	128	45	456	83



2LB. Steel No. 2 Carbonized at 1,650°.

1st quench from furnace at 1,650° (Scler. hardness = *H*; Brin. hardness = *I*). 2d quench at 1,400° (Scler. hardness = *J*; Brin. hardness = *K*).



2LS. Steel No. 2 Carbonized at 1,650°.

(Scler. hardness = *F*; Brin. hardness = *G*). Quenched at 1,400° (Scler. hardness = *J*; Brin. hardness = *K*).

2LD.

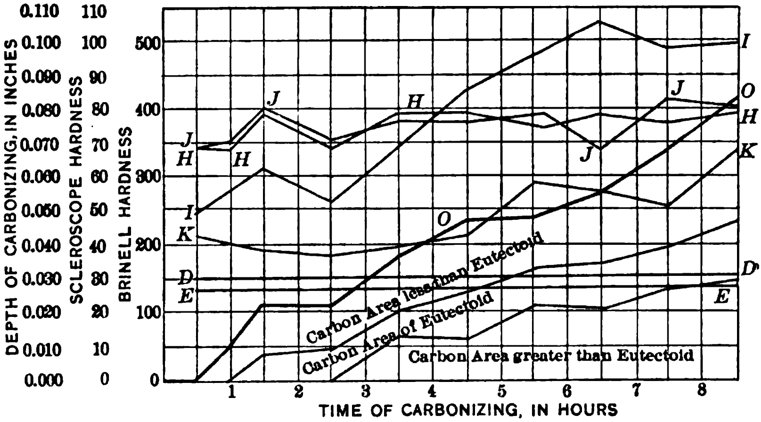
Steel No. 2. Carbonized at 1,650°. 1st Quench at 1,650°; 2d Quench at 1,400°.

No.	Original.		1st Quench.		2d Quench.	
	Brin.	Scle.	Brin.	Scle.	Brin.	Scle.
1	128	......	248	68	211	68
2	130	......	271	68	201	70
3	135	......	304	78	188	78
4	126	......	260	68	183	70
5	135	......	344	78	196	76
6	136	......	426	78	218	76
7	135	......	470	74	375	78
8	136	......	520	76	268	62
9	131	......	486	75	251	82
10	136	......	496	78	331	80

2HB.

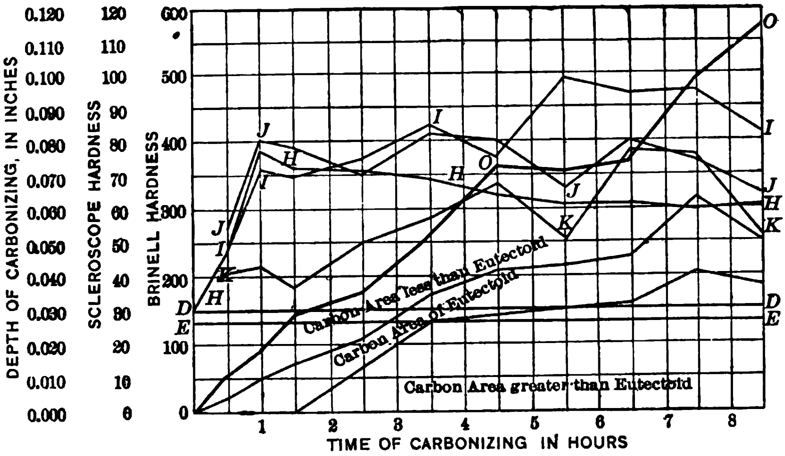
Steel No. 2. Carbonized at 1,750°. 1st Quench from Furnace at 1,750°; 2d Quench at 1,400°.

No.	Original.		1st Quench.		2d Quench.	
	Brin.	Scle.	Brin.	Scle.	Brin.	Scle.
1	128	......	232	46	205	53
2	130	......	358	78	218	80
3	128	......	344	72	185	78
4	130	......	372	71	248	70
5	128	......	425	69	285	82
6	130	......	372	64	338	88
7	131	......	496	61	256	66
8	131	......	470	61	388	80
9	137	......	480	60	379	75
10	135	......	414	61	263	63



2LD. Steel No. 2 Carbonized at 1,650°.

1st quench at 1,650° (Scler. hardness = *H;* Brin. hardness = *I*). 2d quench at 1,400° (Scler. hardness = *J;* Brin. hardness = *K*).



2HB. Steel No. 2 Carbonized at 1,750°.

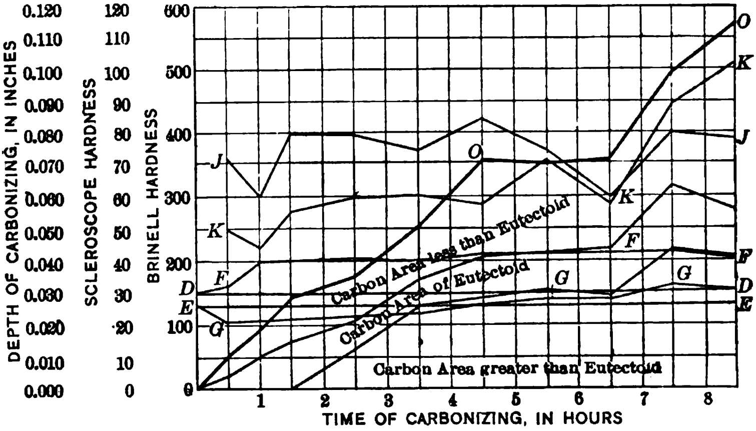
1st quench from furnace at 1,750° (Scler. hardness = *H;* Brin. hardness = *I*). 2d quench at 1,400° (Scler. hardness = *J;* Brin. hardness = *K*).

2HS.
Steel No. 2. Carbonized at 1,750°.
1st Quench at 1,400°

No.	Original.		After Carb.		1st Quench.	
	Brin.	Scle.	Brin.	Scle.	Brin.	Scle.
1	133	30	106	32	248	72
2	135	31	106	40	218	60
3	128	32	110	40	279	60
4	129	30	110	41	302	80
5	131	30	117	40	302	75
6	134	30	131	41	288	85
7	131	31	140	42	358	75
8	128	29	142	42	288	60
9	138	30	160	42	447	80
10	138	29	149	40	508	78

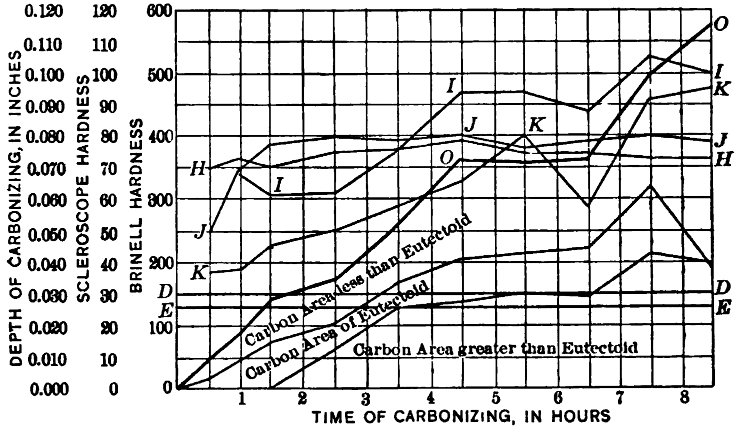
2HD.
Steel No. 2. Carbonized at 1,750°. 1st Quench ot 1,650°; 2d Quench at 1,400°.

No.	Original.		1st Quench.		2d Quench.	
	Brin.	Scle.	Brin.	Scle.	Brin.	Scle.
1	135	28	425	70	186	48
2	133	24	341	72	190	70
3	136	27	306	70	224	78
4	135	30	310	75	344	80
5	126	32	372	75	288	78
6	129	35	470	78	324	80
7	131	35	470	74	399	76
8	133	37	433	74	280	78
9	129	30	526	72	457	80
10	130	29	496	72	470	78



2HS. Steel No. 2 Carbonized at 1,750°.

Scler. hardness = *F;* Brin. hardness = *G.* 1st quench at 1,400° (Scler. hardness = *J;* Brin. hardness = *K*).



2HD. Steel No. 2 Carbonized at 1,750°.

1st quench at 1,650° (Scler. hardness = *H;* Brin. hardness = *I*). 2d quench at 1,400° (Scler. hardness = *J;* Brin. hardness = *K*).

8LB.

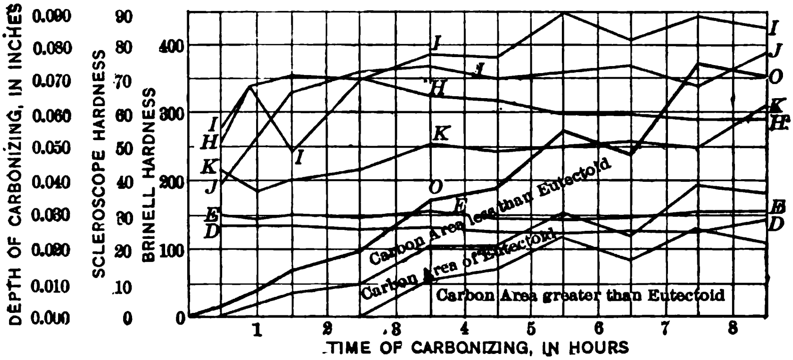
Steel No. 8. Carbonized at 1,650°, 1st Quench from Furnace at 1,650°; 2d Quench at 1,325°.

No.	Original.		1st Quench.		2d Quench.	
	Brin.	Scle.	Brin.	Scle.	Brin.	Scle.
1	151	27	285	53	213	40
2	144	27	338	68	199	53
3	149	26	246	71	203	65
4	149	26	352	70	218	72
5	154	25	388	65	255	74
6	144	25	382	64	242	70
7	144	25	447	60	251	72
8	147	25	407	60	260	74
9	154	25	447	58	248	68
10	154	26	425	58	312	78

8LS.

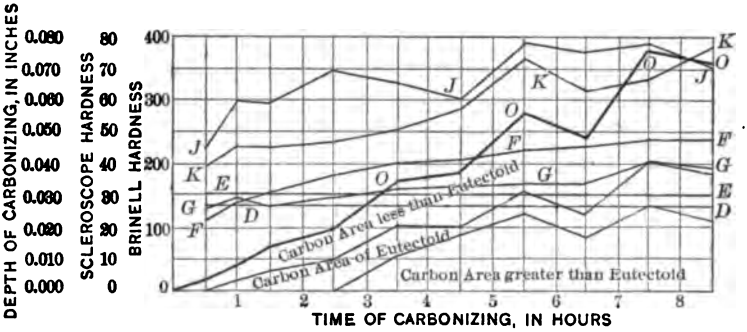
Steel No. 8. Carbonized at 1,650°. 1st Quench at 1,325°.

No.	Original.		After Carb.		1st Quench	
	Brin.	Scle.	Brin.	Scle.	Brin.	Scle.
1	153	......	131	23	198	45
2	154	......	144	28	226	60
3	154	......	131	31	224	60
4	155	......	149	36	235	70
5	151	......	161	41	251	65
6	149	......	161	41	288	60
7	149	......	168	44	366	78
8	149	......	167	45	313	75
9	149	......	198	47	324	77
10	150	......	190	47	379	70



8LB. Steel No. 8 Carbonized at 1,650°.

1st quench from furnace at 1,650° (Scler. hardness = *H;* Brin. hardness = *I*). 2d quench at 1,325° (Scler. hardness = *J;* Brin. hardness = *K*).



8LS. Steel No. 8 Carbonized at 1,650°.

Scler. hardness = *F;* Brin. hardness = *G.* Quenched at 1,325°. (Scler. hardness = *J;* Brin. hardness = *K*).

8LD.

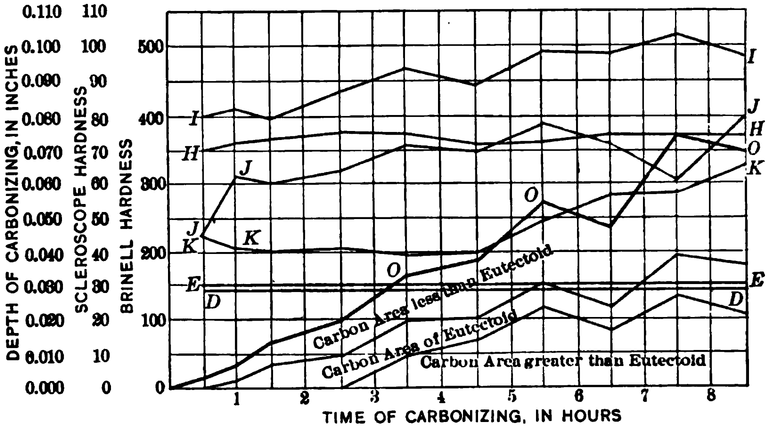
Steel No. 8. Carbonized at 1,650°. 1st Quench at 1,575°; 2d Quench at 1,325°.

No.	Original.		1st Quench.		2d Quench.	
	Brin.	Scle.	Brin.	Scle.	Brin.	Scle.
1	151	25	396	70	221	46
2	151	28	407	72	201	62
3	154	28	398	73	201	60
4	155	28	438	75	206	64
5	160	28	470	75	196	72
6	151	28	447	72	201	70
7	153	27	496	72	256	78
8	151	28	496	75	288	70
9	153	29	526	75	292	62
10	153	29	496	70	330	80

8HB.

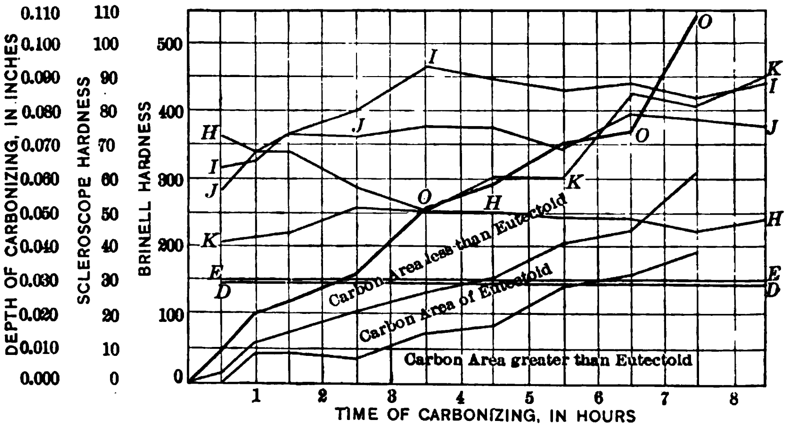
Steel No. 8. Carbonized at 1,750°. 1st Quench from Furnace at 1,750°: 2d Quench at 1,325°.

No.	Original.		1st Quench.		2d Quench.	
	Brin.	Scle.	Brin.	Scle.	Brin.	Scle.
1	154	32	315	72	203	56
2	147	30	326	68	211	68
3	142	29	366	68	218	73
4	139	30	398	57	255	72
5	147	28	466	50	251	75
6	142	30	447	50	298	75
7	139	30	425	48	300	68
8	139	30	438	48	425	78
9	144	32	418	44	407	77
10	147	34	438	47	447	75



8LD. Steel No. 8 Carbonized at 1,650°.

1st quench at 1,575° (Scler. hardness = *H;* Brin. hardness = *I*). 2d quench at 1,325° (Scler. hardness = *J;* Brin. hardness = *K*).



8HB. Steel No. 8 Carbonized at 1,750°.

1st quench from furnace at 1,750° (Scler. hardness = *H;* Brin. hardness = *I*). 2d quench at 1,325° (Scler. hardness = *J;* Brin. hardness = *K*).

8HS.

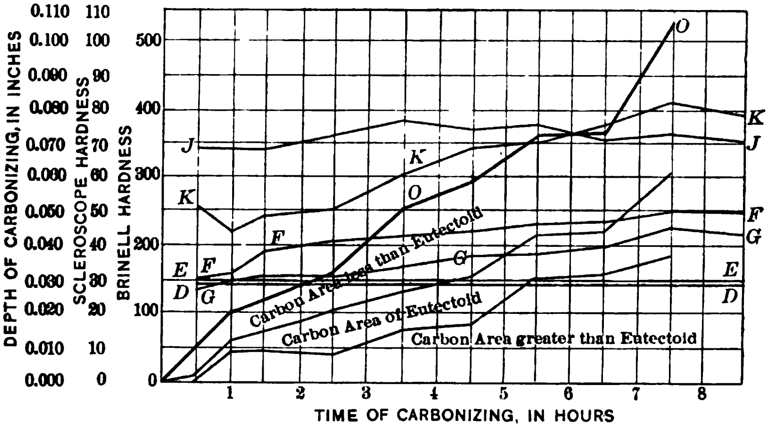
Steel No. 8. Carbonized at 1,750°.
1st Quench at 1,325°.

No.	Original.		After Carb.		1st Quench.	
	Brin.	Scle.	Brin.	Scle.	Brin.	Scle.
1	154	29	135	30	256	68
2	145	29	148	32	218	68
3	151	28	154	38	242	68
4	162	31	154	41	251	72
5	151	30	168	42	302	76
6	154	31	182	44	239	73
7	157	31	186	46	350	75
8	151	29	194	46	372	70
9	155	30	224	50	407	72
10	149	30	212	47	389	70

8HD.

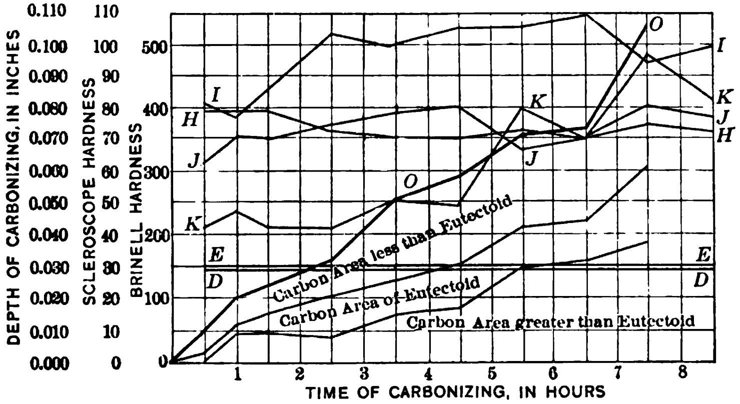
Steel No. 8. Carbonized at 1,750°. 1st Quench.
at 1,575°; 2d Quench at 1,325°.

No.	Original.		1st Quench.		2d Quench,	
	Brin.	Scle.	Brin.	Scle.	Brin.	Scle.
1	146	......	407	78	208	62
2	149	......	382	78	233	70
3	153	......	425	78	208	70
4	155	......	508	72	208	74
5	151	......	496	70	251	78
6	154	......	526	70	242	80
7	158	......	526	72	396	66
8	148	......	546	70	349	70
9	151	......	470	75	485	80
10	149	......	496	72	407	76



8HS. Steel No. 8 Carbonized at 1,750°.

(Scler. hardness = *F;* Brin. hardness = *G*). Quenched at 1,325° (Scler. hardness = *J;* Brin. hardness = *K*).



8HD. Steel No. 8 Carbonized at 1,750°.

1st quench at 1,575° (Scler. hardness = *H;* Brin. hardness = *I*). 2d quench at 1,325° (Scler. hardness = *J;* Brin. hardness = *K*).

DISCUSSION OF RESULTS.

Allowing for a few discrepancies, the results show a consistent uniformity in demonstrating certain facts. I will discuss these in detail:

Original Scleroscope and Brinell Hardness.

These hardnesses are designated by the letters *D* and *E* respectively on all the diagrams. On 2LB and 8LB are plotted the individual hardnesses of the original steel before carbonizing. It will be noted that these lines are practically horizontal, for which reason the original hardness on the other diagrams was taken as the average of all the pieces, and therefore the lines are drawn horizontally.

Depth of Carbonization.

This is plotted in the three areas as previously described and printed on the drawings. The line *O* denotes the maximum depth of carbonization. Under like conditions of carbonization, the rate of penetration in the carbon-steel was 1.07 times the rate in the nickel-steel at the low (1,650°) temperature. At the high temperature (1,750°) this rate for the carbon was 1.15 times that for nickel.

These carbonization-depth curves show the characteristic flattening out and subsequent rise after the initial period of carbonization.

The average rate of penetration of the carbon-steel at the high temperature was 1.48 times the rate at the low temperature, while for the nickel-steel the figure was 1.41, showing the tremendous influence of temperature.

Hardness after Carbonizing and before Quenching.

See 2HS, 2LS, 8HS and 8LS. The most apparent fact which we note here, is the higher scleroscope reading after carbonizing compared with the reading before carbonizing, while the corresponding Brinell readings are but slightly higher in grade No. 8 and even lower in grade No. 2. This may be readily explained. The scleroscope method, being influenced largely by the surface-conditions, here measures the increased hardness due to the higher carbon. It will be noted that there is a sharp rise in this curve at the beginning until the surface-carbonization is greater than 0.90 per cent. and from here on the increase in hardness is more gradual. The Brinell hardness, being more a function of the core than of the surface, is actually made less for the lower depths of carbon, due to the soft annealing produced by the slow cooling in the carbonizing-boxes. However, as the depth of carbonization increases, this Brinell reading is more and more influenced by it and less by the conditions of the core. A careful consideration of the facts thus brought out will show that the No. 8 steel was better annealed originally than No. 2.

Hardness after Quenching from Carbonizing-Box.

See **2HB, 2LB, 8HB, 8LB.** The scleroscope readings show a gradual decrease in hardness with increased depth in carbon, after the first hour of carbonization. For both grades the steel carbonized at the higher temperature shows the greater decrease in hardness with increasing depth of case; and No. 8, for the same temperature of carbonization, shows a lower hardness than No. 2. The explanation is, that, with increasing depths of carbonization the percentage of carbon in the surface becomes higher; in quenching from this high temperature the austenite-forming tendencies increase with increasing depth; and the higher the percentage of austenite, the lower the hardness. For the same steel, the higher temperature of carbonization produces more cementite and consequently austenite; and likewise, at the same temperature, the carbon-steel produces more cementite than the nickel-steel. This explanation can be verified by comparing the scleroscope hardnesses with those produced when the steel is quenched at a lower temperature and the excess of cementite is allowed to remain out of solution.

Let us now note the Brinell hardnesses corresponding to the above scleroscope hardnesses. These show some slight irregularities, undoubtedly due to differences in cooling-rate caused by mechanical difficulties of quenching from the large carbonizing-boxes. However, the curves show an increase in hardness from 4 to 6 hr. and then the hardness becomes practically stationary. This is in direct contrast to the scleroscope hardness. The probabilities are that these Brinell readings would have begun to decrease if longer length of time had been used. This is actually shown in **2HB** and **8HB.** The explanation of this Brinell hardness is, that the influence of the core decreases while the influence of the case increases as the depth of the carbon becomes greater; therefore, as soon as the depth becomes great enough to cause the core-influence to be negligible, the hardness ceases to rise and then decreases, for the same reason as explained in the discussion of the scleroscope readings.

Hardness after First Quench on Double-Quenched Specimens.

See **2LD, 2HD, 8LD, 8HD.** The scleroscope readings after this quench are practically horizontal. The gradual reduction in hardness shown in the higher quench from the box is absent. It will be noted that this hardness is considerably higher than that obtained upon quenching direct from the box, by reason of the weaker austenite-forming tendencies. The Brinell hardness shows the same increase as for those quenched from the box, and also the tendency to begin dropping towards the end. The actual Brinell hardness is also about the same as that obtained when quenching from the box.

Hardness after Second Quench.

See **2LD, 2HD, 8LD, 8HD.** We will now discuss those cooled in the box and then receiving a double quench. Let us first consider what we should expect from a theoretical standpoint. For surfaces of less than 0.90 per cent. of carbon, we should expect a lower hardness than when quenched from a higher temperature, by reason of the fact that an excess of ferrite is now present. In proportion as a higher percentage of carbon than 0.90 is obtained, we should expect a harder surface, due to the fact that we have allowed an excess of cementite to appear. A critical study of the curves indicates that this is the tendency, although the difference is slight. Now, considering the Brinell hardness after the second quench, we here note a very remarkable dropping off in hardness. The hardness at the lower depths of carbonization is approaching the original hardness, and with increasing depth of carbon it shows a decided increase. This is due to the fact that at the low temperature of quenching the core is no longer in the martensitic condition and therefore does not resist the penetration of the Brinell ball; but as the depth of carbon increases the core exerts less and less influence, and the case exerts more, with a consequent rise in the hardness. It will be noted that the hardness does not reach the hardness of the first quench except for the higher heats of carbonization. It is interesting to note that we have here all the data necessary for a determination of the depth of carbonization at which the condition of the core ceases to have any influence upon the Brinell hardness. This is approximately at the point where the Brinell reading on the second quench is the same as that on the first quench. This is about 0.11 in. deep.

The hardness on the second quench of the steel quenched from the box is similar to that of the second quench of the tests just considered.

Hardness after a Single Low Heat.

See 2LS, 2HS, 8LS, 8HS. The scleroscope line here exhibits the same characteristics as previously explained for low heats, except for some irregularities caused by the fact that the cementite here occurs in large masses rather than in disseminated grains. The Brinell line is also similar to that obtained on the second quench of the double heat, with the exception that on the low heats of carbonization it rises faster and to a greater figure than on the double quench. This is undoubtedly an influence of the core, although it is hard to explain unless it is caused by the difference in the condition of dissemination of the ferrite and martensite in the two cases.

The knowledge gained by the above research-work has formed the basis of a system of testing carbonized parts for depth of carbonization and efficiency of the heat-treatment.

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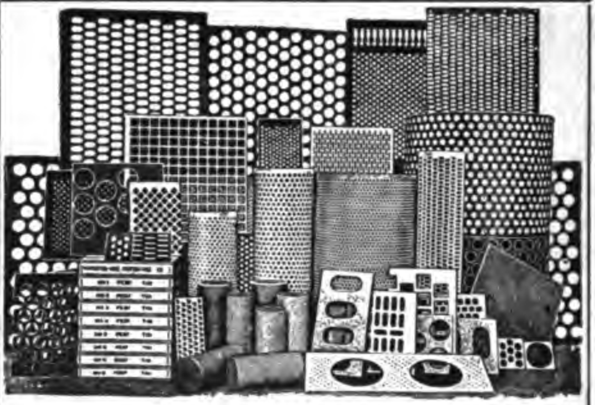

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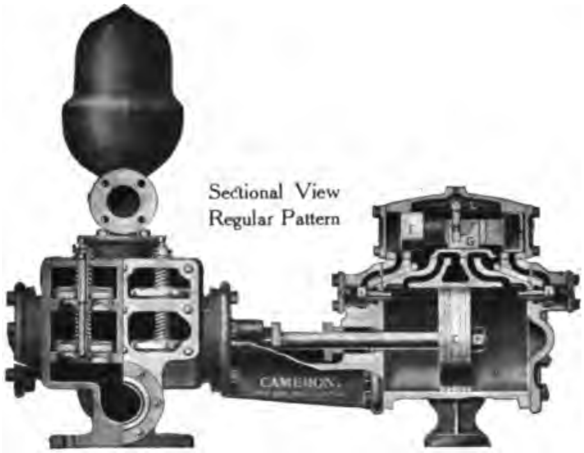

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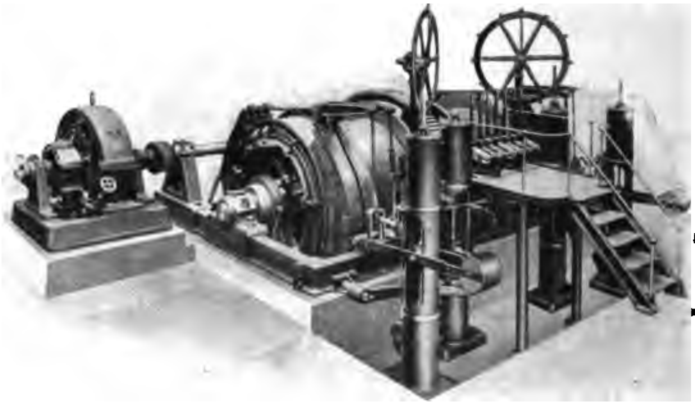

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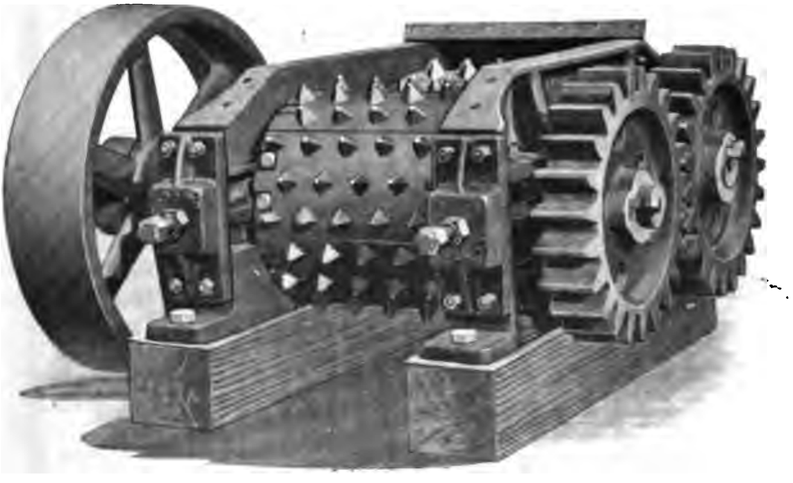

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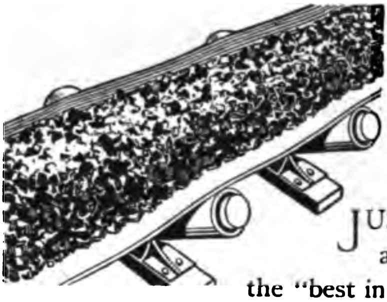

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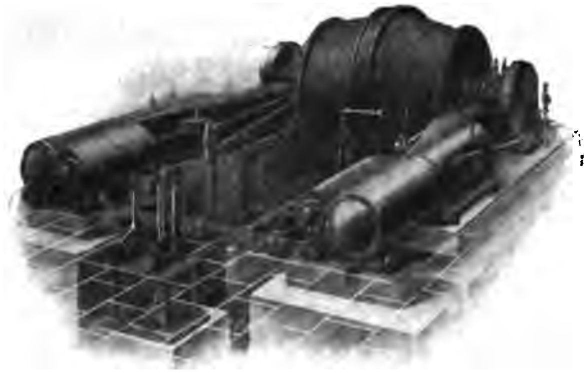

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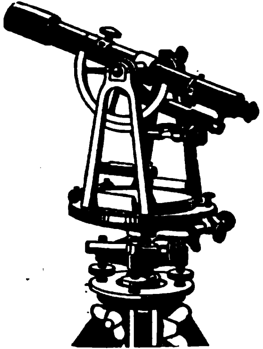

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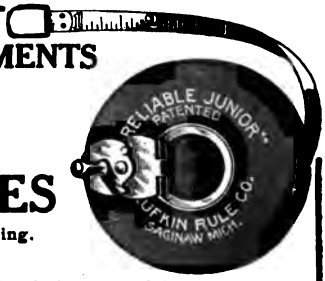
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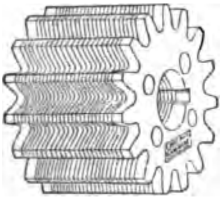

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CONTENTS:

SECTION I.—INSTITUTE ANNOUNCEMENTS.

PAGE
List of Officers and Committees, ... ii
Institute Announcements, ... v
Library, ... xv
Membership, ... xx

SECTION II.—PAPERS AND DISCUSSIONS.

Notes on Ruff's Carbon-Iron Equilibrium Diagram. By Henry M. Howe, ... 1181
The Wood Flotation Process. By Henry E. Wood, ... 1227
Notes on Titanium and on the Cleansing Effect of Titanium on Cast-Iron. By Bradley Stoughton, ... 1245
Surveying and Sampling Diamond-Drill Holes. By E. E. White, . 1277
The Manufacture of Coke. By W. H. Blauvelt, ... 1299
The Manufacture of Coke. By F. E. Lucas, ... 1315
Blowing-In a Blast-Furnace. By R. H. Sweetser, ... 1327
Note on the Case-Hardening of Special Steels. By Albert Sauveur and G. A. Reinhardt, ... 1335
The Alundum Extraction-Thimble Used in the Determination of Copper. By L. W. Bahney, ... 1343
Proceedings of the One Hundred and Third Meeting, Cleveland, Ohio, October, 1912, ... 1345

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TABLE OF CONTENTS.

SECTION I. INSTITUTE ANNOUNCEMENTS.

PAGE
List of Officers and Committees for the Year Ending February, 1913, . ii
Institute Announcements—CIRCULAR OF NOMINATING COMMITTEE, v
Library, ... xv
Membership, ... xx

SECTION II. TECHNICAL PAPERS.

[The American Institute of Mining Engineers does not assume responsibility for any statement of fact or opinion advanced in its papers or discussions.]

No. 1. Notes on Ruff's Carbon-Iron Equilibrium Diagram. By HENRY M. HOWE, ... 1181
No. 2. The Wood Flotation Process. By HENRY E. WOOD, ... 1227
No. 3. Notes on Titanium and on the Cleansing Effect of Titanium on Cast-Iron. By BRADLEY STOUGHTON, ... 1245
No. 4. Surveying and Sampling Diamond-Drill Holes. By E. E. WHITE, 1277
No. 5. The Manufacture of Coke. By W. H. BLAUVELT, ... 1299
No. 6. The Manufacture of Coke. By F. E. LUCAS, ... 1315
No. 7. Blowing-In a Blast-Furnace. By R. H. SWEETSER, ... 1327
No. 8. Note on the Case-Hardening of Special Steels. By ALBERT SAUVEUR and G. A. REINHARDT, ... 1335
No. 9. The Alundum Extraction-Thimble Used in the Determination of Copper. By L. W. BAHNEY, ... 1343
No. 10. Proceedings of the One Hundred and Third Meeting, Cleveland, Ohio, October, 1912, ... 1345

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INSTITUTE ANNOUNCEMENTS.

Special Notice.

The *Bulletin* is now entered at the Post Office at Second-Class Postage rate of one cent per pound, and in order to preserve this privilege it is necessary that the dues of members be paid within four months of January 1 of each year. If the dues are not paid within the period mentioned, a member's name must be removed from the regular subscription-list and the *Bulletin* mailed at the transient second-class postage rate of one cent for each four ounces or fraction thereof, prepaid by stamps affixed. It is therefore earnestly requested that dues be paid promptly—otherwise the Institute will be put to additional expense of postage and to added labor in removing and replacing names from the regular list, and in maintaining an additional separate mailing-list.

Nominations for Officers.

The undersigned, having been appointed a Committee to nominate officers for the Institute for the year 1913, desire that every member participate in the work of this Committee by offering suggestions and proposing names for the various offices, thereby assisting in obtaining the best ticket that can be recommended.

The officers to be elected at the Annual Meeting in February are, a President, three Vice-Presidents, three Councilors, three Directors, and a Secretary.

All suggestions and names with recommendations should be sent to the Chairman of the Committee, Philip N. Moore, 611 Merchants-Laclede Building, St. Louis, Mo., and should be received by him before Dec. 1, 1912.

The list of officers whose terms expire is given herewith, as well as the names of the officers whose terms do not expire.

The Committee desires to propose for the different offices men who will be representative and who will also represent the various sections of the United States, but, at the same time, it must be remembered that it is necessary so to constitute the Board that a sufficient number of the officers will have their headquarters in New York or vicinity, so that a quorum for meetings can be secured.

Of the officers (excluding President and Secretary) whose terms expire, four are from New York and five from elsewhere in the United States; of those whose terms do not expire, ten are from New York and its vicinity and five from elsewhere in the United States and Canada.

As already stated, it is hoped that the members of the Institute will assist the Committee to the utmost by making suggestions and proposals as requested above.

PHILIP N. MOORE, *Chairman.*
JOHN BIRKINBINE.
D. W. BRUNTON.
SEELEY W. MUDD.
CHARLES P. PERIN.
ARTHUR L. WALKER.

Officers Whose Terms Expire in February, 1913.

PRESIDENT OF THE COUNCIL.

JAMES F. KEMP	NEW YORK, N. Y.

VICE-PRESIDENTS OF THE COUNCIL.

S. B. CHRISTY	BERKELEY, CAL.
R. V. NORRIS	WILKES-BARRE, PA.
GARDNER F. WILLIAMS	WASHINGTON, D. C.

COUNCILORS.

ROBERT E. JENNINGS	NEW YORK, N. Y.
WILLIAM KELLY	VULCAN, MICH.
HENNEN JENNINGS	WASHINGTON, D. C.

SECRETARY OF THE COUNCIL.

JOSEPH STRUTHERS, 29 W. 39th St	NEW YORK, N. Y.

DIRECTORS.

JAMES GAYLEY	NEW YORK, N. Y.
CHARLES KIRCHHOFF	NEW YORK, N. Y.
FRANK LYMAN	NEW YORK, N. Y.

Officers Whose Terms Do Not Expire in February, 1913.

VICE-PRESIDENTS OF THE COUNCIL.

KARL EILERS	NEW YORK, N. Y.
WALDEMAR LINDGREN	WASHINGTON, D. C.
BENJAMIN B. THAYER	NEW YORK, N. Y.

(Term expires February, 1914.)

COUNCILORS.

A. E. CARLTON	CRIPPLE CREEK, COLO.
W. J. OLCOTT	DULUTH, MINN.
E. L. YOUNG	NEW YORK, N. Y.

(Term expires February, 1914.)

JOHN H. JANEWAY, JR	NEW YORK, N. Y.
SIDNEY J. JENNINGS	NEW YORK, N. Y.
JOSEPH W. RICHARDS	SOUTH BETHLEHEM, PA.

(Term expires February, 1915.)

DIRECTORS.

JAMES DOUGLAS, JAMES F. KEMP, ALBERT R. LEDOUX.

(Term expires February, 1914.)

EDMUND B. KIRBY, CHARLES F. RAND, GEORGE C. STONE.

(Term expires February, 1915.)

Iron and Steel Division.

COMMITTEE.

CHARLES KIRCHHOFF, *Chairman.*
CHARLES F. RAND, *Vice-Chairman.*
BRADLEY STOUGHTON, *Secretary*, 165 Broadway, New York.

John Birkinbine,
James Gayley,
Henry D. Hibbard,
Henry M. Howe,
Charles K. Leith,
Robert W. Hunt,
Julian Kennedy,
Richard Moldenke,
Joseph W. Richards,
Albert Sauveur,
F. W. C. Schniewind,
A. A. Stevenson,
Felix A. Vogel,
Leonard Waldo,
William R. Webster.

At the recent 103d meeting of the Institute, in Cleveland, Oct. 28–31, there were 29 papers dealing with iron or steel, extending over a wide range of subjects, from iron-ore mining to scientific investigations of theoretical principles underlying metallurgical practice. All of these papers were available in printed or mimeographed form, so that those who desired to prepare discussion in advance could obtain copies. Copies of many of the papers were also distributed in advance to about a hundred members who had asked for them in respose to the Committee's circular stating that they would be available. The Committee takes this opportunity to explain that lateness of authors in sending in manuscript or in returning corrected proof, put such a strain on the resources of the printer at the very last minute, that advance copies could not be sent out as early as had been hoped, and mimeographing had to be resorted to in the case of about half a dozen manuscripts.

Fifteen papers were presented to the technical sessions by their authors or author's representative, and fourteen of these were discussed. Only lack of time prevented a longer discussion of some papers, and it is the desire of the Committee to promote discussion by correspondence until all available data on each subject have been obtained from members of the Institute. Copies of any of the papers will be sent to any member upon request to the Secretary of the Committee, and members are earnestly requested to forward all discussion which will advance the subject. Even items of seemingly minor interest, by the insight they give into practice, or by their cumulative effect, often prove to be of great value. Where the discussion evoked seems to warrant it, the Committee will request that some papers be brought up again at later meetings of the Institute where the author may meet others in further debate.

A significant fact has been brought out by R. H. Sweetser, of Columbus, O., and that is that to blast-furnace managers, the paper of greatest importance was not a paper on iron, but was that on copper-slags by Prof. Charles H. Fulton, of the Case School of Applied Science, Cleveland, O. Mr. Sweetser holds that the facts brought out will lead to the most valuable series of investigations in the metallurgical side of blast-furnace practice attempted since the days of Lowthian Bell. The fact that copper metallurgists have advanced further than those engaged in the production of iron, shows how much the co-operation of allied and associated branches is needed, through the intermediary of the general organization of the Institute.

A nucleus of iron and steel papers for the next meeting of the

Institute has already been promised, and the Committee will endeavor to set a high standard for all papers accepted, although, at the same time, encouraging members who have not yet contributed to bring forward their views and results of experience.

BRADLEY STOUGHTON, *Secretary*,
165 Broadway, New York, N. Y.

New York Local Section.

Executive Committee.

GEORGE F. KUNZ, *Chairman.*
E. GYBBON SPILSBURY, *Vice-Chairman.*
THOMAS ROBINS.
H. B. SEAMAN.
LOUIS D. HUNTOON, *Secretary-Treasurer.*

The second meeting of the 1912–13 season of the New York Section of the American Institute of Mining Engineers was held at the Engineering Societies Building, 29 West 39th Street, New York, on Wednesday evening, Oct. 23, at 8.30 p.m.

Prof. William Campbell, Ph. D., Sc. D., of Columbia University, delivered an illustrated lecture on The Microscopic Examination of Opaque Materials. This lecture was announced by the mailing of 950 reply post-cards to members of the Section and members of the Institute residing in Connecticut, New York, New Jersey, Washingington, D. C., and Pennsylvania. The replies indicated an attendance of 86. Although the weather was most disagreeable there was an attendance of 70.

As an introduction, Professor Campbell described the preparation of specimens for examination, followed by a description of the main types of microscope, and illustrated by a special stand, together with a very ingenious method of mounting specimens by the use of a short cylinder closed at one end and containing a pyramid of wax. The specimen to be mounted is placed on the table with its polished surface down and the wax is pressed against the specimen. By this method the specimen is imbedded in the wax, and the cylinder containing the specimen can be placed on the stand under the microscope in a horizontal position and at the required distance from the focal lens, thereby avoiding the necessity of having a mechanical stage and adjustments, which are very expensive.

Following this introduction, the lecture, which was illustrated by a large number of beautiful photographs, covered the structure of metals, grains and dendrites, effects of strain, slip-lines, the effects of annealing, and grain-growth and twinning. The structure of alloys was systematically explained in detail, including the copper-nickel series, brasses and the effects of corrosion, solders, ternary alloys, such as lead-tin-antimony and copper-tin-antimony, and zinc-rich alloys. The various changes in the solid state were illustrated by the bronzes, iron, and steel. Starting with wrought-iron, the latter series was exemplified by structural steel, good and bad; so-called "recrystallization" and fatigue of axles; castings; segregation and other faults in rails; tool-steel and faulty treatment; and the cast-irons; such as car-wheel iron. Next ores were shown from the Cobalt; Cochise district, Arizona; Butte; Sudbury; Lowell, Mass.; and elsewhere, and examples of secondary enrichment were shown. Lastly, metallurgical products were illustrated by the copper-copper oxide series, showing blister, underpoled, tough-pitch and over-poled copper, iron- and copper-mattes, nickel-mattes of various grades and compositions, both Sudbury and European; also blast-furnace and refinery speisses.

To those acquainted with the subject the lecture showed remarkable skill on the part of Professor Campbell in his work, and to those less acquainted with the subject, it was a revelation of the progress of modern methods of metallography. The lecture was thoroughly appreciated.

Prof. James F. Kemp, President of the Institute, and Prof. Waldemar Lindgren, of the U. S. Geological Survey, discussed the subject in detail.

After the discussions the Chairman introduced Prof. Eugene Oberhommer, Professor of Geography in the University of Vienna, who described his Western trip with the members of the International Congress of Applied Chemistry.

The members and guests adjourned to the offices of the Institute, where refreshments were served by the Reception Committee, spending a pleasant hour in social intercourse.

BUDGET.

At the meeting of the Executive Committee of the New York Section held Oct. 3, 1912, a resolution was passed directing the Secretary to ask the members of the Section to contribute to the budget of $1,000 required to meet the estimated expenses of the Section and to invite members of the Institute residing in Connecticut, New York, New Jersey, Pennsylvania, and Washington, D. C., to become associate members of the Section. The resident members were asked to contribute not over $10 each, and the associate members $2. Letters in response indorsing the organization of local sections and offering to contribute further if necessary, and the large number of checks received, both from the resident and associate members, have been most gratifying. The contributions up to and including Oct. 30, 1912, amounted to a total of $650, made up as follows:

Members.

No.	Amount.	Total.	Grand Total.
4	$2.00	$8.00	
7	3.00	21.00	
2	4.00	8.00	
35	5.00	175.00	
36	10.00	360.00	
1	25.00	25.00	
			$597.00

Associate Members.

No.	Amount.	Total.	Grand Total.
9	$2.00	$18.00	
3	5.00	15.00	
2	10.00	20.00	
			53.00
			$650.00

With such support from the members at large the New York Section should have a most successful year.

LECTURES.

The following lectures have been promised for the year:
Mr. N. H. Darton, Structural Geology, on Nov. 22.
Mr. F. G. Cottrell, Metallurgy of Copper.
Mr. H. V. Winchell, Mining Law.
Mr. L. D. Ricketts, Lixiviation of Oxidized Ores.

Arrangements are being made for lectures on The Flushing of Mines, Placer and Hydraulic Mining, Cyaniding, Ore-Deposits of Ontario, and The Practice on the Rand.

LOUIS D. HUNTOON, *Secretary-Treasurer,*
165 Broadway, New York, N. Y.

Boston Local Section.

ROBERT H. RICHARDS, *Chairman.*
ALBERT SAUVEUR, *Vice-Chairman.*
AUGUSTUS H. EUSTIS, *Secretary-Treasurer.*

The fifth meeting of the Boston Section was held at the University Club, Monday evening, Oct. 7, 1912, about fifteen members attending at dinner, and others, including a few members of Affiliated Student Societies, coming in after the dinner. The members present expressed themselves as well pleased with the plan of dining together before the meeting, and voted that a dinner be arranged before the next meeting. The meeting was called to order by the Chairman, Prof. R. H. Richards. The minutes of the previous meeting were read and accepted.

The Chairman then called on Prof. Henry L. Smyth, who gave an interesting informal talk on The Iron Bearing Rocks of the Lake Superior Region. Professor Smyth commenced his address by stating that applied geology had been spoken of as the art of making an intelligent guess from incomplete data. He spoke of the great difficulty in determining the relative age of the various strata in the different iron-ranges, there being no fossils to assist in such a classification. He went on to show that the facts developed by diamond-drilling for the iron-mines with which he is associated, seemed to indicate that some of the conclusions stated in the Geological Survey monograph, recently published on this section, were erroneous.

The discussion following the paper was largely led by Professor Lane, formerly State Geologist of Michigan; and the meeting adjourned in high spirits.

The sixth meeting of the Boston Local Section was called for Monday evening, Nov. 4, 1912, at the Technology Club, 83 Newbury Street. The program comprised an informal talk by Prof. Raphael Pumpelly on Excavations in Turkestan.

A. H. EUSTIS, *Secretary-Treasurer,*
131 State St., Boston, Mass.

Adjourned Annual Business Meeting, Oct. 7, 1912.

The following is an abstract of the Minutes of the Adjourned Annual Business Meeting of the Institute, adjourned first to June 3, 1912, and later, by mutual consent of the Board of Directors and the Committee of Five, to Monday, Oct. 7, 1912, pursuant to regular notice. (There was no meeting on June 3, 1912).

The meeting was called to order by President James F. Kemp at the Institute Headquarters in New York City.

The Secretary reported that 53 members were present in person, 1021 by proxy to Messrs. Hunt, Eilers and Kirchhoff, and 275 by proxy to Messrs. Corning and Finlay. Total, 1,349.

The minutes of the Annual Business Meeting of the Institute held Feb. 20, 1912, were read and approved.

Prof. Joseph W. Richards, *Chairman* of the Special Committee of the Board and Council, which presented certain proposed amendments to the Constitution on Feb. 20, 1912, action on which had been postponed to the Adjourned Business Meeting, withdrew these proposed amendments.

Certain proposed amendments to the Constitution were presented by Messrs. C. R. Corning and George C. Stone,* to be voted upon at the Annual Business Meeting of the Institute, Feb. 18, 1913.

Prof. Joseph W. Richards presented proposed amendments to two articles of the Constitution, to be voted upon at the Annual Business Meeting of the Institute, Feb. 18, 1913.*

Prof. Joseph W. Richards, in behalf of the Special Committee of the Directors and Council appointed to confer with the Committee of Five, advised that conferences had been held, both by these Committees, and by the Directors and Council and the Committee of Five, at a special meeting, Oct. 4, 1912, and that while agreement had been reached in some instances, the majority of the Directors and Council did not approve of the articles proposed by Messrs. Corning and Stone, as follows:

"If any Member, Junior Member or Associate is in arrears for four months, the *Bulletin* shall no longer be sent to him and he shall be notified by the Secretary of the Institute that no publications will be sent him until his arrears are paid.

"The professional, technical, scientific and social interests of the Institute shall be committed to the supervision of the Council composed of a Chairman of the Council, a First Vice-Chairman of the Council, six Vice-Chairmen and nine Councilors, who shall be elected from among the members and associates of the Institute in the manner hereinafter provided. Members of the Council may or may not be members of the Board of Directors.

"Section 3, which deals with the election at each annual meeting of the President, Vice-President and Secretary of the Institute, change President and Vice-President to be Chairman and Vice-Chairman.

"Nominations may also be made *viva voce* at the annual meeting by any properly qualified voter.

"The Directors shall appoint a business manager who shall, under the instructions of the Board, have charge of the business affairs of the Institute. The duties of the Business Manager shall be such as the Board may assign to him and must not include incurring of expense not provided for by appropriations nor the disbursal of funds except as provided for by the Constitution. The salary of the Manager shall be fixed by the Board, and he shall hold office during its pleasure."

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* Printed in Announcement No. 8, and sent to the membership under date of Oct. 15, 1912.

Dr. A. R. Ledoux moved that these and any other amendments to the Constitution which may be proposed to-day be printed and sent to the membership at large for information.* He suggested that an informal discussion could be held at a session of the Cleveland meeting,† and that said amendments in final form could be again submitted at an adjournment of the present adjourned meeting, for vote at the Annual Business Meeting, Feb. 18, 1913. After general discussion by Messrs. A. R. Ledoux, C. Kirchhoff, C. R. Corning, P. N. Moore, Joseph W. Richards, C. P. Perin, the motion was carried.

Certain proposed amendments to the By-Laws* were presented by Messrs. C. R. Corning and George C. Stone, and after general discussion by Messrs. Hennen Jennings, P. N. Moore, C. R. Corning, Sidney J. Jennings, C. F. Rand, George C. Stone, Joseph W. Richards, and E. W. Parker, it was

Voted: To print and send to the membership at large these proposed amendments to the By-Laws; also a brief circular letter summarizing the chief points at issue.*

On motion by P. N. Moore, seconded by C. R. Corning, it was

Voted: That it is the sense of this meeting that the dual system of government should be abolished; and to that end the Committee on Amendments to the Constitution (Messrs. Richards, Kirchhoff and Rand) be requested to take action; and that the Council be directed to take action toward unifying the two governing bodies of the Institute; or else toward defining with absolute clearness the restrictive functions of each body.

Mr. C. F. Rand presented the following resolutions:

Resolved, That the report of the Committee of Five, which has already been received and distributed to members, be now filed with the accompanying documents in the archives of the Institute, and further

Resolved, That the thanks of the Institute are due and are hereby extended to C. R. Corning, *Chairman;* J. F. Kemp, G. C. Stone, W. H. Nichols, Jr., and A. R. Ledoux, the members of the Committee of Five, for their services in the preparation of their report.

Resolved, That the Committee be, and the same is, hereby discharged.

These resolutions were seconded by C. Kirchhoff as one of the representatives of more than one thousand members of the Institute; and the vote in favor was unanimous.

The meeting was adjourned until Tuesday, Nov. 12, 1912, at 10 a. m., at Institute Headquarters in New York City.

An abstract of the minutes of the business session of the Institute, held at Cleveland, Oct. 30, 1912, is given in the Proceedings of the One Hundred and Third Meeting, Cleveland, Ohio, pp. 1345 to 1355 of this issue of the *Bulletin.*

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* Printed in Announcement No. 8, and sent to the membership under date of Oct. 15, 1912.

† See pp. 1347 to 1349 of this *Bulletin.*

A New Volume on Ore-Deposits.

Ore-Deposits—a continuation of the "Posepny" Volume.

Comprising Papers Descriptive of Ore-Deposits and Discussions of Their Origin, Edited, with an Introduction, by Dr. S. F. Emmons. The volume contains also a Biographical Notice of Dr. Emmons by his associate and friend, Dr. George F. Becker, and a comprehensive Bibliographical Index of the Science of Ore-Deposits, prepared by Prof. John D. Irving, of the Sheffield Scientific School of Yale University. Dr. Emmons had finished his editorial work and written his Introduction before his lamented death in 1910; and the Volume contains his last words upon the subject to which he had given the work of his life, and on which he was justly regarded as the foremost authority.

Further details concerning time of publication, price, etc., will follow.

Library Research-Work.

The attention of members of the Institute is again directed to the research-work done by the librarian and his assistants, which should attract special attention from those members who have no access to the literature of subjects in which they may be interested.

During the year 1911 there were 143 searches made for members and non-members of the Founder Societies, and copies of the references have been preserved for the use of others. This work has been largely based on requests sent in by mail, from Japan, South Africa, Mexico, Canada, and England, as well as from different parts of the United States. The Librarian is confident that if it were more widely known that the library is equipped to undertake researches, the demand would increase beyond the ability of the present force to handle it. The library receives more than 700 technical periodicals which are available through the indexes for this special purpose.

Back Volumes of the Transactions.

The Board of Directors has authorized the following offers of sets of back volumes of the *Transactions*, at considerably reduced prices, to Members, Libraries, and Scientific Societies:

		Per Set.
I.	Five volumes, bound in half-morocco, from No. 36 (1906) to No. 40 (1910), ...	$20
II.	Ten volumes, bound in half-morocco, from No. 31 (1902) to No. 40 (1910), including Mexican Volume, . . .	35
III.	Twenty volumes, bound in half-morocco, from No. 21 (1893) to No. 40 (1910), ...	50
IV.	Thirty volumes, bound in half-morocco, from No. 11 (1883) to No. 40 (1910), ...	60
V.	Thirty-nine volumes, bound in half-morocco, from No. 1 (1873) to No. 40 (1910), with the exception of No. 10 (1882), but including index for Volumes Nos. 1 to 35, and Nos. 36 to 40, ...	75
VI.	Nine volumes, bound in half-morocco, from No. 1 (1873) to No. 9 (1881), ...	25

Applications should be addressed to Joseph Struthers, Secretary, 29 West 39th Street, New York, N. Y.

The Emmons Research Fellowship of Economic Geology.

The Committee named below has been formed by friends of Samuel Franklin Emmons, late of the United States Geological Survey, to consider the best method of perpetuating his name. It has been decided that the memorial to him shall take the shape of a Research Fellowship, to be known as the Samuel Franklin Emmons Research Fellowship of Economic Geology, which is to be administered by Prof. James F. Kemp, of Columbia University, New York. Subscriptions are invited by his friends to this fund, which the Committee has fixed at $25,000.

Members of the Institute who desire to contribute to this fund will please communicate with the Treasurer, Benjamin B. Lawrence, 60 Wall Street, New York.

The Committee consists of the following:

GEORGE OTIS SMITH, Director, U. S. Geological Survey, Washington, D. C.
H. L. SMYTH, Harvard University, Cambridge, Mass.
JAMES DOUGLAS, 99 John Street, New York, N. Y.
J. A. HOLMES, Director, Bureau of Mines, Washington, D. C.
JAMES F. KEMP, Columbia University, New York, N. Y.
F. W. BRADLEY, San Francisco, Cal.
J. PARKE CHANNING, 42 Broadway, New York, N. Y.
SEELEY W. MUDD, 1001 Central Building, Los Angeles, Cal.
D. W. BRUNTON, Denver, Colo.
H. FOSTER BAIN, 420 Market Street, San Francisco, Cal.
T. A. RICKARD, London, England.
B. B. LAWRENCE, 60 Wall Street, New York, N. Y.

LIBRARY.

AMERICAN INSTITUTE OF ELECTRICAL ENGINEERS.
AMERICAN SOCIETY OF MECHANICAL ENGINEERS.
AMERICAN INSTITUTE OF MINING ENGINEERS.
UNITED ENGINEERING SOCIETY.

WILLIAM P. CUTTER, Librarian.

The libraries of the above-named Societies are open from 9 A.M. to 9 P.M. on all week-days, except holidays, from September 1 to June 30, and from 9 A.M. to 6 P.M. during July and August.

The Library contains about 50,000 volumes, including sets of technical periodicals and the publications of scientific and technical societies.

The members of the Institute, with few exceptions, are by the very nature of their profession forced to spend a large portion of their time in localities isolated from sources of information. To such members the Library can render valuable service through correspondence, and letters requesting information will receive special attention. The Library is prepared to furnish references and copies of articles on mining and metallurgical subjects; to determine, if possible, the existence of mining-maps, and to furnish general information as to the geology and mineral resources of all countries as far as these resources are known and published.

It is hoped that the members of the Institute will avail themselves freely of this special service. The Library will welcome inquiries on engineering subjects, and furnish information as far as such information is to be obtained.

All communications should be made as definite as possible so that the information received may be what is desired and not include collateral matter which may not be of interest. In this way the time spent in searching for such collateral matter will be saved, and as a result the information will be sent more promptly and in more usable shape.

The members of the Institute can be of service to the Library by forwarding copies of mining-reports, maps privately issued, and similar material, which will be classified, indexed, and made available to other members.

Suggestions for additions to the Library, either by purchase or personal solicitation as gifts, will be welcomed. It is hoped that members while in the city will use the Library freely, and assurance is given that most careful service will be rendered to them.

Library Accessions.

Oct. 1 to Oct. 31, 1912.

[Copies of the list of additions to the Libraries of the American Society of Mechanical Engineers and the American Institute of Electrical Engineers can be obtained on application to the Secretary of the American Institute of Mining Engineers.]

METALLOGRAPHY OF IRON AND STEEL. By Albert Sauveur. Cambridge, 1912. Price, $6. (Gift of Sauveur & Boylston.)

A notice of this volume appeared in *Bulletin* No. 70, October, 1912, under the erroneous title, "Metallurgy of Iron and Steel."

ANTIMONY MINING INDUSTRY AND THE DISTRIBUTION OF ANTIMONY ORES IN NEW SOUTH WALES. Sydney, 1912. (Exchange.)

ARMSTRONG COLLEGE. CALENDAR, 1912-13. London, 1912. (Gift of Armstrong College.)

BELGIUM. MINISTÈRE DE L'INTÉRIEUR. Annuaire statistique de la Belgique et du Congo Belge. Tome XLII., 1911. Bruxelles, 1912. (Gift of Ministère de l'Intérieur.)

BILDER UND ZAHLEN AUS DEM BERGBAUE OSTERREICHS. Festgabe an die teilnehmer des Allgemeinen Bergmannstages. Wien, 1912. (Gift of O. H. Hahn.)

[NOTE.—This handsome quarto is a souvenir, prepared by a Committee for presentation to the delegates attending the General Miners' Parliament, held this year in Vienna. It presents in official tables and graphic diagrams the extent of the Austrian production of coal and lignite, coke, iron, gold, silver, mercury, lead, zinc, bismuth, antimony, uranium, sulphur, graphite, asphalt, salt, and petroleum, together with data of wages, accidents, etc.; and it is adorned with a large number of excellent half-tone pictures of mining localities and works, which make it as beautiful as it is useful.—R. W. R.]

BROKEN HILL SOUTH SILVER MINING CO. Reports, Statements of Accounts, etc., for half-year ended June 30, 1912. Melbourne, 1912. (Gift of W. E. Wainwright.)

CANADA. MINES DEPARTMENT. Geology and Ore Deposits of Phœnix Boundary District, British Columbia. (Memoir No. 21.) Ottawa, 1912. (Exchange.)

——— Southern Vancouver Island. (Memoir No. 13.) Ottawa, 1912. (Exchange.)

——— Report on the Building and Ornamental Stones of Canada. Vol. 1. Ottawa, 1912. (Exchange.)

——— Report on the Utilization of Peat Fuel for the Production of Power. (No. 154.) Ottawa, 1912. (Exchange.)

——— Summary Report of the Geological Survey Branch, 1911. Ottawa, 1912. (Exchange.)

CANAL RECORD. Vol. 5, 1911-1912. Ancon, Canal Zone, 1912. (Gift of Isthmian Canal Commission.)

CARNEGIE LIBRARY OF PITTSBURGH. Classified Catalogue, Part 1, 1907-1911. Pittsburgh, 1912. (Exchange.)

COMITÉ DES FORGES DE FRANCE. Annuaire, 1912-13. Paris, 1912. (Exchange.)

CONSOLIDATED MINING & SMELTING CO. OF CANADA, LTD. Report of the Directors, 1912. Toronto, 1912. (Gift of Consolidated Mining & Smelting Co. of Canada, Ltd.)

COPPER MINES OF THE URAL REGION. By H. W. Turner. N. p., n. d. (Gift.)

DIGEST OF WORKMEN'S COMPENSATION LAWS, 1912. A reference work for the legislator, lawyer, insurance expert, employer and employee. N.p., 1912. (Gift of National Association of Manufacturers.) Price, 25 cents.)

[NOTE.—This useful pamphlet, prepared by the National Association of Manufacturers and the Manufacturers' Association of Connecticut, gives concise but complete details of the workmen's compensation laws of fourteen States. This information is so grouped as to give of each particular subject involved a comparative view, showing how each State has treated it. As the title-page intimates, such a comprehensive exhibit is indispensable to legislators, lawyers, insurance experts, employers and employees. The various statutes here analyzed and compared are so many experiments, the results of which should lead to the general adoption of a wise and practicable, as well as just and humane, system of distributing among the three parties concerned—the employer, the employee and the community—the burden of industrial accidents to health, limb or life.—R. W. R.]

DAS ERDÖL UND SEINE VERWANDTEN. Ed. 3. By Hans von Höfer. Braunschweig, 1912. (Gift of Author.)

[NOTE.—It is hardly necessary to say more of this book than that it is a third and enlarged edition of the well-known classic work with which Councilor Höfer's name is associated, and which constitutes perhaps the most valuable of his many noteworthy contributions to technical literature. It is dedicated "in full gratitude" to four societies of which the author is an honorary member; and this Institute heads the list. The second edition appeared in 1906, eighteen years after the first. This one follows after only five years—a striking evidence of the increased interest in its subject and the rapid growth of knowledge and theory concerning petroleum and the bitumens. The most striking and important part of this great work is the wonderfully comprehensive, critical discussion of the numerous theories of the origin of petroleum and the formation of oil-deposits. This has been brought up to date by the inclusion of such modern investigations as those of Engler. The author still adheres to the view which he has done so much to expound and defend —that petroleum is derived from organic material, chiefly of animal character, though he does not exclude the co-operation of certain oil- or wax-bearing plants, such as diatomacea and fungi. His masterly summary of the arguments of all parties is a history as well as a judgment, and, taken together with his abundant references to original sources, practically exhausts the literature of the subject.—R. W. R.]

GAZETEER OF SURFACE WATERS OF CALIFORNIA. Part II. San Joaquin River Basin. Water Supply Paper No. 296, U. S. Geological Survey. Washington, 1912. (Exchange.)

HETCH HETCHY WATER SUPPLY FOR SAN FRANCISCO, 1912. Report by John R. Freeman. San Francisco, 1912. Price, $3. (Gift of John R. Freeman.)

INDIA MINES DEPARTMENT. Report of the Chief Inspector of Mines, 1911. Calcutta, 1912. (Exchange.)

IRON INDUSTRY IN RUSSIA. Economical Statistical Sketch. (In Russian.) St. Petersburg, 1911. (Purchase.)

IRON ORES OF MISSOURI. (Missouri Bureau of Geology and Mines. Second Series, Vol. X.) Jefferson City, 1912. (Exchange.)

JAARVERSLAG DER TINWINNING OP BANKA OVER HET EXPLOITATIEJAAR, 1910–1911. Batavia, 1912. (Exchange.)

KAURI GUM MINING IN NEW ZEALAND. By R. A. F. Penrose. Reprint from *Journal of Geology*, Vol. XX., No. 1. N.p., n.d. (Gift of Author.)

KEWEENAW SERIES OF MICHIGAN. Vols. 1–2. By A. C. Lane. Publication 6, Geological Series 4, Michigan Geological and Biological Survey. Lansing, 1911. (Exchange.)

MAINE SOCIETY OF CIVIL ENGINEERS. Constitution, Proceedings and List of Members, No. 1, 1911. Augusta, 1911. (Gift of Maine Society of Civil Engineers.)

MICHIGAN GEOLOGICAL AND BIOLOGICAL SURVEY. Publication 6, Geological Series 4, Vols. I.–II.; Publication 8, Geological Series 6. Lansing, 1911–12. (Exchange.)

MINERAL RESOURCES OF MICHIGAN, WITH STATISTICAL TABLES OF PRODUCTION AND VALUE OF MINERAL PRODUCTS FOR 1910 AND PRIOR YEARS. Publication 8, Geological Series 6, Michigan Geological and Biological Survey. Lansing, 1912. (Exchange.)

MINING WORLD INDEX OF CURRENT LITERATURE. Vol. I., 1912. Chicago, 1912. (Gift of Mining World Co.)

[NOTE.—There is no doubt of the usefulness of such a bibliography as this. The only question that can arise concerning it is, whether it has been well done; and this will be answered in the affirmative by any one who will examine the book. The contents are arranged under convenient headings, in three parts, entitled respectively Geographics, Ores and Mineral Products, and Technology. The entries show the author's name ; the title ; the periodical, page and date ; the approximate number of words ; and the price. There is also an alphabetical index of authors and another of subjects.—R. W. R.]

MISSOURI BUREAU OF GEOLOGY AND MINES. Second Series. Vol. X. Jefferson City, 1912. (Exchange.)

NEW SOUTH WALES—MINES DEPARTMENT. Mineral Resources. No. 16. Sydney, 1912. (Exchange.)

NEW ZEALAND—GEOLOGICAL SURVEY BRANCH. Bulletin No. 14. Wellington, 1912. (Exchange.)

NORTH OF ENGLAND INSTITUTE OF MINING AND MECHANICAL ENGINEERS. Annual Report of the Council and Accounts for the Year 1911–12. Newcastle-upon Tyne, 1912. (Exchange.)

ONTARIO BUREAU OF MINES. Report. Vol. XXI. Part 2. Toronto, 1912. (Gift of Ontario Bureau of Mines.)

ORIENTAL CONSOLIDATED MINING CO. Report, 1912. New York, 1912. (Gift of Oriental Consolidated Mining Co.)

PRINCETON ENGINEERING ASSOCIATION. Officers, Constitution, and List of Members, 1912–13. N. p., n. d. (Gift of Princeton Engineering Association.)

READING IRON CO. Wrought Iron Pipe vs. Steel Pipe. Testimony and Arguments. Reading, 1912. (Gift of Reading Iron Co.)

RESULTS OF SPIRIT LEVELING IN NEW YORK, 1906–1911, INCLUSIVE. Bulletin No. 514, U. S. Geological Survey. Washington, 1912. (Exchange.)

RESULTS OF SPIRIT LEVELING IN TENNESSEE, 1910–11. Bulletin No. 519, U. S. Geological Survey. Washington, 1912. (Exchange.)

RIVISTA DEL SERVIZIO MINERARIO NEL, 1911. Roma, 1912. (Exchange.)

STANDARD SPECIFICATIONS, UNIFORM PROCEDURE, AND FORMS RELATING TO COAL. (City of New York, Department of Water Supply, Gas, and Electricity. Bureau of Economy and Efficiency, Bulletin No. 2, Feb., 1912.) New York, 1912. (Gift of J. L. Pultz.)

SURFACE WATER SUPPLY OF THE UNITED STATES, 1910. Part XI. Pacific Coast in California. (Water Supply Paper No. 291, U. S. Geological Survey.) Washington, 1912. (Exchange.)

SYDNEY UNIVERSITY ENGINEERING SOCIETY. Proceedings. Vol. XVI. Sydney, 1911. (Exchange.)

UNION DES INGÉNIEURS SORTIS DES ECOLES SPÉCIALES DE LOUVAIN. Bulletin. 2d Ser., Vol. VI., Book 4, 1912. Bruxelles, 1912. (Gift.)

UNION OF SOUTH AFRICA. MINES DEPARTMENT. Annual Report. Part I. and II. Pretoria, 1912. (Gift of Mines Department of Union of South Africa.)

U. S. BUREAU OF MINES. Bulletin. Nos. 44, 47. Washington, 1912. (Exchange.)

——— Technical Paper. Nos. 22, 24, 26. Washington, 1912. (Exchange.)

U. S. COAST AND GEODETIC SURVEY. Effect of Topography and Isostatic Compensation upon the Intensity of Gravity. (Second paper.) By Wm. Bowie. Washington, 1912. (Exchange.)

U. S. GEOLOGICAL SURVEY. Bulletin. Nos. 514, 519. Washington, 1912. (Exchange.)

——— Geologic Atlas of the United States. Folio No. 183 (Llano-Burnet, Texas). Washington, 1912. (Exchange.)

——— Water Supply Paper. Nos. 291, 296. Washington, 1912. (Exchange.)

WESTERN AUSTRALIA. MINES DEPARTMENT. Report, 1911. Perth, 1912. (Exchange.)

GIFT OF HILL PUBLISHING CO.

AMERICAN ASSOCIATION OF COMMERCE AND TRADE IN BERLIN. Yearbook, 1912. Berlin, 1912.

CANADA. CUSTOMS DEPARTMENT. Report, 1912. Ottawa, 1912.

——— Trade and navigation. Unrevised monthly statements of imports entered for consumption, July, 1912. Ottawa, 1912.

INDIA. SURVEY. On the Origin of the Himalaya Mountains. Calcutta, 1912.

NORWAY. STATISTISKE CENTRALBYRAA. Norges Handel, 1911. Kristiania, 1912.

PAINT CREEK SITUATION. A review from the Operator's Standpoint. N.d.

REVISED MINING LAWS OF THE STATE OF IOWA, May 15, 1911. Des Moines, 1911.

ROBINSON, NEIL. "West Virginia on the Brink of a Labor Struggle." Charleston, 1912.

SVERIGES OFFICIELLA STATISTIK. Bergshanftering ar 1911. Stockholm, 1912.

TRADE CATALOGUES.

ASBESTOS PROTECTED METAL CO., Beaver Falls, Pa.
  Bulletin No. 16. A. P. M. roofing and siding.
  Bulletin No. 51. Asbestosteel concrete roof and floor.

CHICAGO PNEUMATIC TOOL CO., Chicago, Ill. Bulletin No. 121. Pneumatic rammers and foundry equipment, 1912. 7 pages.

JEFFREY MANUFACTURING CO., Columbus, Ohio. Bulletin No. 19 A. Ball-bearing motors for mine service. 4 pages.

A. LESHEN & SONS ROPE CO., St. Louis, Mo. Leschen's Hercules. Oct., 1912.

McKIERNAN-TERRY DRILL CO., New York, N. Y. "Busy Bee" hammer drills. 15 pages.

MINE & SMELTER SUPPLY CO., Denver, Colo. Bulletin No. 25. No. 6 Wilfley concentrator. 19 pages.

STEPHENS-ADAMSON MFG. CO., Aurora, Ill. Labor saver, Oct., 1912.

UNIVERSAL CRUSHER CO., Cedar Rapids, Iowa. Some facts about the Velten Universal Crusher.

WEBSTER MFG. CO., Chicago, Ill. Webster method. Sept., 1912.

United Engineering Society Library.

EDWARD HENRY HARRIMAN. By John Muir. New York, 1912. (Gift of Doubleday, Page & Co.)

PUBLISHERS' TRADE LIST ANNUAL, 1912. New York, 1912. (Purchase.)

UNITED STATES CATALOG BOOKS IN PRINT JANUARY 1, 1912. Ed. 3. Minneapolis-New York, H. W. Wilson Co., 1912. (Purchase.)

TRADE CATALOGUES.

NEW YORK AIR BRAKE CO., New York, N. Y. Circular No. 49 A. Locomotive foundation driver brakes. Schedule and prices.

PARKESBURG IRON CO., Parkesburg, Pa. Descriptive Catalogue. 43 pages.

MEMBERSHIP.

New Members.

The following list comprises the names of those persons elected as members who accepted election during the month of October, 1912:

Members.

Abbott, Robert R., Met. Engr..........Peerless Motor Car Co., Cleveland, Ohio.
Adami, Charles J., Min. Engr..Rivermines, Mo.
Ammon, Mark A., Met. Engr............Peerless Motor Car Co., Cleveland, Ohio.
Barnard, Enoch A., Asst. Testing Engr., Washoe Smelter, P. O. Box 275, Anaconda, Mont.
Barnes, Walter A., Min. Engr........... Utah Metal Mining Co., Tooele, Utah.
Barratt, Sydney A., Supt., Ponupo Manganese Co., El Cuero Iron Mines, Santiago de Cuba, Cuba.
Best, William N., Engr. in Caloric.................11 Broadway, New York, N. Y.
Borowsky, Abraham G., Secy., Penn Rivet Mfg. Co., Inc., 11th and Cambria Sts., Philadelphia, Pa.
Broden, Albert................ Supt. Blast Furn., Reading Iron Co., Reading, Pa.
Brown, Halcombe J., Vice-Prest. and Secy., Crown Gypsum Co., 31 Main St., Buffalo, N. Y.
Burley, Curtis R., Chem...Gem, Idaho.
Callaway, Fred. W., Min. Engr., Bunker Hill & Sullivan M. & C. Co., Kellogg, Idaho.
Cantley, Charles L., Asst. to Genl. Mgr., Nova Scotia Steel & Coal Co., Ltd., New Glasgow, N. S., Canada.
Cousin, Robert, Min. Engr., 95 Muirkirk Rd., Catford, London, S. E., England.
Coyle, John A., Supt., Hess Steel Castings Co., P. O. Box 476, Bridgeton, N. J.
Dickerman, William C., Vice-Prest., Am. Car & Fdy. Co., 165 Broadway, New York, N. Y.
Dows, David, Ore Concentration, American Grondal Co., 50 Church St., New York, N. Y.
Garnier, Thomas R., Min. Engr..............1735 Mission St., So. Pasadena, Cal.
Goltra, Edward F., Pres., Missouri Iron Co....La Salle Bldg., St. Louis, Mo.
Grace, Eugene G., Vice-Prest. and Genl. Mgr., Bethlehem Steel Co., So. Bethlehem, Pa.
Greenough, W. Earl, Mining...Wallace, Idaho.
Hall, John H., Met. Engr............Taylor Iron & Steel Co., Highbridge, N. J.
Hartman, William E., Mng. Engr., H. Koppers Co., 5 Wabash Ave., Chicago, Ill.
Hunnings, Samuel V., Chem. and Engr. of Tests, American Locomotive Co., Schenectady, N. Y.
Jayne, Amel R., Min. Engr.....................319 New York Blk., Seattle, Wash.
Kinney, Harry D., Min. Engr.............Utah Metal Mining Co., Tooele, Utah.
Kirkland, Thornton C., Asst. Mgr., Cia. Met. y Refinadora del Pacifico, S. A., Fundicion, Son., Mexico.
Klepinger, John H., Asst. Supt., B. & M. Redn. Wks., Anaconda Copper Mining Co., Great Falls, Mont.
Klopstock, Paul, Genl. Mgr., Gold Note Min. & Mil. Co., Kennedy, Nev.
Landgrebe, Karl L., Supt. Blast Furn., Tenn. C. I. & R. R. Co., Ensley, Ala.
Lindau, Sam P., Chem...Swansea, Ariz.
Linhardt, Harvey W., Blast Furn. Engr., Columbus Iron & Steel Co., Columbus, Ohio.
Lipscomb, Robert A., Furnace Supt.......... ....Low Moor, Va.
McKee, William C., Asst. Supt. Blast Furn., 3154 Aldis Ave., Indiana Harbor, Ind.

McLeod, Howard D., Mgr., Min. Dept., Seattle Const. & Dry Dock Co., Seattle, Wash.
Mooney, James D., Sales Engr............................Leonard Hotel, Butte, Mont.
Morgan, George H., Min. Engr...........................P. O. Box 795, Globe, Ariz.
Morrow, Bayard S., Min. Engr...........................P. O. Box 102, Gem, Idaho.
Morse, George H., Min. Engr...............................Pocahontas, Alta., Canada.
Newhall, Henry B., Jr., Contracting............ . .90 West St., New York, N. Y.
Oberg, Anton C., Asst. Chief Min. Engr., Arthur Iron Mining Co., Hibbing, Minn.
Parrock, H. P., Engr..............................Lumen Bearing Co., Buffalo, N. Y.
Perry, William C., Min. Engr.........Davis Coal & Coke Co., Cumberland, Md.
Preston, Veryl........Prest. Eastern Steel Co., 60 Broadway, New York, N. Y.
Prickett, William C., Min. and Mech. Engr., 1212 N. 12th Ave., Birmingham, Ala.
Roden, Benjamin F...............................Prest. Roden Coal Co., Marvel, Ala.
Sargent, Fitz-William, Chief Engr., Am. Brake Shoe & Fdy. Co., Mahwah, N. J.
Saulles, Charles A. H. de, Am. Smelt. & Refin. Co., 165 Broadway, New York, N. Y.
Shimer, William R., Met.................Bethlehem Steel Co., So. Bethlehem, Pa.
Snyder, Henry S., Vice-Prest............Bethlehem Steel Co., So. Bethlehem, Pa.
Thomas, Perry C., Min. Engr..Macdonald, W. Va.
Thompson, Frank E., Supt........................Cambria Steel Co., Johnstown, Pa.
Toy, Francis L., Met................................927 Maryland Ave., Pittsburg, Pa.
Wilfley, Clifford R., Mgr................................Bristow Mine, Ouray, Colo.
Wood, Frederick W., Prest..............Maryland Steel Co., Sparrows Point, Md.
Young, Charles D., Engr. of Tests...............Pennsylvania R. R., Altoona, Pa.

Associates.

Crowther, John S., Jr., Blast Furn. Blower, 420 St. Louis St., Toledo, Ohio.
Eppley, Marion, Banker and Broker...............80 Broadway, New York, N. Y.
Eshleman, S. Kendrick, Jr., Steel and Iron, P. O. Box 16, Leaman Place, Lancaster Co., Pa.
Graef, Harry C., Prest., Montana & Mexico Mining Co., 29 Broadway, New York, N. Y.
Murphy, Edward M., Elect. Engr...............Hecla Mining Co., Burke, Idaho.
Roeser, Ernest B., Min. Engr..................................Kanawha Falls, W. Va.
Trotter, Felix F., Min. Engr................ . ...Plaza Hotel, Corey, Ala.
Yang, Cho, Student...............................484 Berkel Ave., So. Bethlehem, Pa.

CANDIDATES FOR MEMBERSHIP.

The following persons have been proposed during the month of October, 1912, for election as members of the Institute. Their names are published for the information of members and associates, from whom the Committee on Membership earnestly invites confidential communications, favorable or unfavorable, concerning these candidates. A sufficient period (varying in the discretion of the Committee, according to the residence of the candidate) will be allowed for the reception of such communications, before any action upon these names by the Committee. After the lapse of this period, the Committee will recommend action by the Council, which has the power of final election.

Members.

George Oliver Deshler, Dunnfield, N. J.
Proposed by N. S. Keith, Edward Hart, W. B. Marquard.

Ernest William Ellis, Anaconda, Mont.
Proposed by C. D. Demond, F. W. C. Whyte, Richard S. McCaffery.

Henry Hess, Bridgeton, N. J.
Proposed by Leonard Waldo, Bradley Stoughton, Louis D. Huntoon.

Joseph Park Hodgson, Bisbee, Ariz.
Proposed by Charles A. Mitke, Max Roesler, Roger T. Pelton.

P. Jansen, Lebong Tandai, Sumatra.
Proposed by W. S. Brown, Charles W. Merrill, H. Foster Bain.

Frederick Gilberts King, San Francisco, Cal.
Proposed by E. B. Kirby, George A. Schroter, P. L. Foster.

Franz Hugo Kleinschmidt, Landore, Ida.
Proposed by Marcus White, J. F. Inglis, Adin P. Tyler.

Frederick Laist, Anaconda, Mont.
Proposed by E. P. Mathewson, William Wraith, C. W. Goodale.

Lloyd Judkins Lathrap, Virginia City, Nev.
Proposed by Whitman Symmes, Emmet D. Boyle, George J. Young.

Philip Jay Lonergan, Jr., Peking, China.
Proposed by George W. Schneider, C. T. Durell, C. L. Colburn.

Carl Herbert Loux, Rossland, B. C., Can.
Proposed by Richard S. McCaffery, Francis Jenkins, C. A. Stewart.

Alexander P. McClure, Cleveland, Ohio.
Proposed by C. B. Murray, J. E. Thropp, Jr., R. H. Sweetser.

Henri Milou, Houlgate, Calvados, France.
Proposed by Auguste J. Rossi, David T. Day, J. William Smith.

Edward Coke Morse, Republic, Wash.
Proposed by S. H. Richardson, Richard Marsh, J. V. Richards, L. K. Armstrong.

Richard Cunningham Patterson, Jr., New York, N. Y.
Proposed by J. F. Kemp, Charles P. Berkey, J. F. McClelland, A. F. Lucas.

Gustav Adolph Reinhardt, Cambridge, Mass.
Proposed by D. A. Lyon, A. W. Smith, Charles H. Fulton.

William Trevor Swoyer, El Paso, Texas.
Proposed by Norval J. Welsh, Kirby Thomas.

Jose Ramon Villalon, Havana, Cuba.
Proposed by F. L. Grammer, Henry S. Drinker, Joseph W. Richards.

Robert T. Walker, Humboldt, Ariz.
Proposed by Arthur H. Wethey, S. R. Trengrove, Harrington Blauvelt.

Yinchang Tsenshan Wang, New York, N. Y.
Proposed by J. F. Kemp, Charles P. Berkey, Henry S. Munroe.

Harold Lee Welsh, Anaconda, Mont.
Proposed by C. D. Demond, William Wraith, C. W. Goodale.

Charles Chambers Woodhouse, Jr., Republic, Wash.
Proposed by Francis A. Thomson, J. V. Richards, L. K. Armstrong.

Associate.

William Clifford Rehfuss, Philadelphia, Pa.
Proposed by J. W. Richards, H. Eckfeldt, H. S. Drinker.

CHANGES OF ADDRESS OF MEMBERS.

The following changes of address of members have been received at the Secretary's office during the month of October, 1912. This list, together with the lists published in *Bulletin* Nos. 63 to 70, March to October, 1912, and the foregoing list of new members, therefore, supplements the annual list of members corrected to Feb. 1, 1912, and brings it up to the date of Nov. 1, 1912.

ADAMS, ARTHUR K., Min. Geol.................................... ....Mogollon, N. M.
ADAMS, JOHN C., Asst. Genl. Supt. of Mines, Anaconda Copper Mining Co., Butte, Mont.
ANDERSON, ALEXANDER......................27 Inverleith Row, Edinburgh, Scotland.
BANKS, HARRY P............................Camp 2, Big Creek P. O., Fresno Co., Cal.

BATES, MOWRY, Bates Bros., Inc., Building Construction 27 William St., New York, N. Y.
BEALL, ALBERT S. E., Hatfield Machinery Co., 306 E. 3d St., Los Angeles, Cal.
BREWER, ROBERT E..........Hinckley Road, Milton, Mass.
BROWNING, WILLIAM C., Gunn-Thompson Co., Newhouse Bldg., Salt Lake City, Utah.
CARR, HENRY C..........Apt. 701, The Woodward, Washington, D. C.
CHADBOURNE, H. W..........14 Wall St., New York, N. Y.
CLARK, CARLE D........Care W. Rowland Cox, 165 Broadway, New York, N. Y.
COLE, T. F..........1701 E. First St., Duluth, Minn.
COOK, EDWARD H..........Birimoa, via Canelas, Durango, Mexico.
CRAUFURD, ALEX. J. F..........P. O. Box 202, Rossland, B. C., Canada.
CRONIN, PAUL B..........Hampton, Iowa.
DRAPER, CARL H., Mesquital Mining Co., Apartado 157, Guadalajara, Jal., Mexico.
ELLIOTT, ROY H., Mine Supt..........Liberty Bell G. M. Co., Tulluride, Colo.
FRANCKLYN, CHARLES G..........519 Cortlandt Ave., New York, N. Y.
FRANKE, EMIL A..........914 Lakeside Pl., Chicago, Ill.
GAMM, HENRY F. E., Chief Engr., Oil Furnace & Engineering Co., 60 Liberty St., New York, N. Y.
GERHARDT, REGINALD B..........2143 Penn St., Harrisburg, Pa.
GOODALE, CHARLES W., Min. Engr., Mgr. B. and M. Dept., Anaconda Copper Mining Co., Butte, Mont.
GRUNSKY, CARL E., JR., Supt......Standard Consolidated Mining Co., Bodie, Cal.
GWINN, JAMES..........Box 421, Kellogg, Idaho.
HAAS, HERBERT..........524 Merchants Exch. Bldg., San Francisco, Cal.
HANCE, JAMES H..........202 Perry St., Elgin, Ill.
HARRIS, ARTHUR L..........Beristain, Pue., Mexico.
HIBBS, JOSEPH G..........613 Pennsylvania Bldg., Philadelphia, Pa.
HUANG, SAOSAN K..........P. O. Box 66, Bethlehem, Pa.
IRWIN, FREDERIC..........2357 W. 31st St., Los Angeles, Cal.
JENNINGS, HENNEN..........2221 Massachusetts Ave., Washington, D. C.
JULIHN, CARL E..........281 Natoma St., San Francisco, Cal.
LAMB, MARK R., Allis-Chalmers Co., Casilla 2653, Santiago, Chile, So. America.
LANDRUM, CHARLES W., Cons. Engr..........602 Keystone Bldg., Joplin, Mo.
LAUBE, FREDERICK E..........1710 Eldridge Ave., Bellingham, Wash.
LEAVELL, JOHN H..........Newhouse Bldg., Salt Lake City, Utah.
LIHME, CHRISTIAN B..........1350 N. State St., Chicago, Ill.
LINDSAY, LYCURGUS..........627 Security Bldg., Los Angeles, Cal.
MCCOLLOM, C. R..........486 California Terrace, Pasadena, Cal.
MALCOLMSON, CHARLES T..........2064 McCormick Bldg., Chicago, Ill.
MEIN, WILLIAM W., Care Canadian Mining & Exploration Co., Ltd., 43 Exchange Place, New York, N. Y.
MELLISS, DAVID E., Cons. Civ. and Min. Engr., P. O. Box V, Mill Valley, Cal.
MILLER, ARTHUR J., American Direct Concentrating Co., 315 South 10th East St., Salt Lake City, Utah.
MOSTOWITSCH, WLADIMIR........Imperial Technological Institute, Tomsk, Siberia.
PARNALL, S. A..........1807 Manhattan Place, Los Angeles, Cal.
PARSONS, CYRIL E..........P. O. Box 94, Salisbury, Rhodesia, So. Africa.
PILL, JOHN R..........Corona, Ala.
PLATE, H. ROBINSON, Min. Engr..........14 Wall St., New York, N. Y.
PRINCE, ERNEST..........1841 Athen St., Boulder, Colo.
REHFUSS, LOUIS A..........Jackson, Cal.
REYNOLDS, GEORGE B., 15 Victoria St., Westminster, London, S. W., England.
SCHUETTENHELM, JOHN B..........Goldfield, Nev.
SHALLCROSS, VINCENT F..........65 St. Georges Terrace, Perth, West Australia.
SHERA, EDWARD L..........421 N. 6th St., Tacoma, Wash.
SHARWOOD, WILLIAM J..........Homestake Mining Co., Lead, S. D.
SHERRERD, ALEX. H..........754 Parker St, Newark, N. J.
SMITH, FREDERICK D., Mgr..........Snow Creek Mine, Greenhorn, Ore.
SMITH, STEWART K..........5541 Maple Ave., St. Louis, Mo.
SOLOMON, TOMAS..........Apartado 39, Pachuca, Hid., Mexico.
STRONG, ARCHIBALD MCC..........530 Union Oil Bldg., Los Angeles, Cal.
TEXTOR, OSCAR..........603 Superior Ave., N. W., Cleveland, Ohio.

Thayer, Reginald H. .... 12 Windsor Terrace, Yonkers, N. Y.
Thomas, Marion L., Min. Engr. .... Manzanillo, Colima, Mexico.
Timm, John M. .... Metzerstrasse 53, Bâle, Switzerland.
Tombo, Carl .... 509 W. 122d St., New York, N. Y.
Trauerman, Carl J. .... Butte Central Copper Co., Butte, Mont.
Truschkoff, Nicolas, Mgr., Eidertinsky Gold Mines, Russian Gold Exploration Co., Orsk, Dept. of Orenburg, Russia.
Turner, Nathaniel P. .... Hotel Camaguey, Camaguey, Cuba.
Tuttle, Arthur L., Genl. Supt., Continental Mining Co., 420 Martin St., San Antonio, Texas.
Tyler, Nat, Jr., Prest., Monarch Road Roller Co., 50 Church St., New York, N. Y.
Van Slyke, Frederick E. .... Care Beckett & Anderson, Glasgow, Scotland.
Warden, Bruce R., Anderson, Warden & Wilkin, Vancouver Block, Vancouver, B. C., Can.
Wedge, Utley, Engr., Sulphuric Acid Expert .... Ardmore, Pa.
Welsh, Norval J. E., Genl. Supt., Jarilla Copper Synd., Inc., Orogrande, N. M.
Wendel, Charles de .... 56 Ave. Hoche, Paris, France.
Wheeler, Archer E., Supt. Redn. Wks., B. and M. Dept., Anaconda Copper Mining Co., Great Falls, Mont.
Williams, Ralph B., Am. Smelt. & Refin. Co., Hacienda Bonanza, Bonanza, Zac., Mexico.
Wilson, George Bennett .... Racine Rubber Co., Racine, Wis.
Wishon, Walter W. .... 1721 S. Vermont Ave., Los Angeles, Cal.
Yeandle, William H., Jr., U. S. Smelt., Ref. & Min. Co., 411 Edificio de la Mutua, Mexico City, Mexico.
Yonge, Allen M .... La Union Mine, Miramar, Costa Rica, C. A.

ADDRESSES OF MEMBERS AND ASSOCIATES WANTED.

Name.	Last Address of Record, from which Mail has been Returned.
Armstrong, Richard E.,	416 E. So. 2d St., Salt Lake City, Utah.
Baxter, Francis K., Jr.,	423 Wells Fargo Bldg., San Francisco, Cal.
Danforth, A. H.,	Cotopaxi, Colo.
David, W. M.,	Care C. G. Hussey & Co., Second Ave., Pittsburg, Pa.
Dyott, George M.,	50 Church St., New York, N. Y.
Eggers, John H., Jr.,	Hughes Creek Mine, Kings River P. O., Cal.
Fergusson, Hugh B.,	216 Loo Bldg., Vancouver, B. C., Canada.
Fitzgerald, Thomas F. M.,	211 Sharon Bldg., Salt Lake City, Utah.
Furness, James W.,	Coffee, Trinity Co., Cal.
Gmelin, Ernest,	Apartado 57, Aguascalientes, Mexico.
Greenley, Louis A.,	Dixie Meadows Mining Co., Prairie City, Ore.
Higgins, Edwin,	Randsburg, Cal.
Hilson, Cleaveland,	Livingston, Mont.
Hollis, R. W.,	Silverton, Colo.
Horschitz, Richard J.,	P. O. Box 453, Haileybury, Ont., Canada.
Johnson, Dion L.,	325 Water St., Pittsburg, Pa.
Lampshire, John O.,	Vulture Mine, Wickenburg, Ariz.
Le Noir, Frank H.,	Box 16, Mt. Bullion, Cal.
McDougall, Wallace D.,	20 Bedford Place, Russell Sq., London, Eng.
Mitinskij, Alexander N.,	V. O. 12, line 33, St. Petersburg, Russia.
Moore, Roy W.,	P. O. Box 48, Velasco, Tex.
Munroe, Martin,	Bengal Coal Co., Murulidih, Mohada, B. N. Ry., Bengal, India.
Nelson, D. W. C.,	Baker City, Ore.
Oldfield, Frank W.,	Mexican Mines Co., Bolaños, Jal., Mexico.
Perks, Harry B.,	419 Board of Trade Bldg., Portland, Ore.
Peterson, Frank,	H. W. Hellman Bldg., Los Angeles, Cal.
Rathborne, Merwyn R. W.,	Amargosa, via Las Vegas, Nev.
Rhew, James W.,	Cia. Minera y Exploradora de Ventanas, S. A., Ventanas, Dur., Mex.
Rickard, Harold,	Foley-O'Brien, Ltd., So. Porcupine, Ont., Canada.
Sheldon, Waldo,	Urique, Chih., Mexico.
Thornton, Edward T.,	Apartado 30, Matehuala, S. L. P., Mexico.

Twynam, Henry, . . . O. K. Copper Mine, Cairns, No. Queensland, Australia.
Vinicombe, Robert E. B., ... Post Restante, Vladikafkas, So. Russia.
Watson, Ralph W., ... Calloo, Utah, Clifton Mail box.
Webster, Erastus H., ... Hotel Cosmopolita, Guadalajara, Jal., Mexico.
White, W. P. . . . . 10 Bernard St., Spokane, Wash.

NECROLOGY.

The deaths of the following members and associate were reported to the Secretary's office during the month of October, 1912:

Date of Election.	Name.	Date of Decease.
1875.	**Attwood, George,	——— —, 1912.
1892.	*Bachman, David I.,	October 8, 1912.
1889.	*McCrery Charles,	——— —, 1912.
1911.	†Weaver, Henry M.,	October 3, 1912.
1886.	*White, Maunsel,	——— —, 1912.

BIOGRAPHICAL NOTICE.

Joseph Charles Kent was born July 17, 1825, in Utica, N. Y., where his father, William St. George Kent, coming from Bath, England, first settled, removing afterwards to New York City. In the latter city, he received his education, which included the study of chemistry under Professor Torrey, of Columbia College. About 1846, he entered the employ of Cooper & Hewitt at the iron-works at Phillipsburg, N. J., of which he became manager five or six years later. The plant and mines of this enterprise were transferred about 1866 to the Andover Iron Co., Mr. Kent continuing to be the manager; and in 1894, he retired, after a continuous and highly successful administration of 40 years.

Mr. Kent, being himself a chemist, was one of the first iron-masters of the old school to realize the advantage of utilizing in blast-furnace practice the analysis of ore, fuel and flux—a thing which we of the next generation can hardly conceive to have been at one time a revolutionary novelty. He also developed from his practical experience several valuable inventions.

One of these was the Kent hot-blast oven, containing circular "U" pipes, with both legs socketed into the same bed-pipe, the latter being provided with diaphragms which freed the circulation through the "siphon" U-pipes. This device furnished greater heating-surface, better circulation, and higher blast-temperature than did the iron-pipe stove previously employed. Of course, it had to yield ultimately to the regenerative hot-blast ovens, producing the much higher temperatures required by modern practice.

Another invention of Mr. Kent's was a blast-furnace hoist, consisting in a cylinder of large dimensions, floating in a cistern, and elevated by the compressed air of the blast, in such a way that (by means of suitable mechanism) a 10-ft. rise of the cylinder effected a 50-ft. rise of the cages carrying the stock. This pioneer improvement was afterwards superseded by J. M. Hartman's pneumatic blast-furnace hoist. Many other leaders of progress have been, like Mr. Kent, overtaken and passed by the advancing army which, for a time, they led; but it is a pleasure, as well as a duty, to recall their useful contributions to the cause.

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* Member. ** Life Member. † Associate.

It was during Mr. Kent's management of the Cooper Iron-works (1856) that the test of the Bessemer process was begun by Cooper & Hewitt at those works. (See the address of Sir James Kitson, in 1890, presenting the Bessemer medal to Abram S. Hewitt, and Mr. Hewitt's reply, *Trans.*, xix., 516, 517. Also my biographical notice of Mr. Hewitt, *Trans.*, xxxiv., 190.) The apparatus constructed for this purpose under Mr. Kent's supervision immediately after the first announcement of Bessemer's invention, seems never to have been put into actual operation, discouraging reports from England having led to the postponement of the plan.

Mr. Kent joined the Institute in 1872, almost immediately after its organization, and remained a loyal and interested member for 40 years. After his retirement, in 1894, he resided at Edgartown, Mass. In many ways, outside his profession, he was widely known and loved as a personal friend, a public-spirited citizen, and an earnest and active Christian. He died at Edgartown Aug. 31, 1912, and was buried at Easton, Pa., across the Delaware river from the works he had so long conducted. The citizens of Phillipsburg and Easton, as well as the local press, gave impressive evidence of their affectionate regard for Mr. Kent, to which I take the liberty of adding my individual testimony of esteem for his character and work, and my sorrow in his departure. R. W. R.

William Arthur Lathrop was born Aug. 4, 1854, at Springville, Susquehanna county, Pa., and after a preliminary education in Springville entered Lehigh University, from which he received in 1875 the degree of Civil Engineer. In June, 1875, he entered, as assistant mining engineer, the service of Irving A. Stearns, with whom he remained for several months, resigning in order to accept a position with the Lehigh Valley R. R. Co., in the Engineering Department, where after a time he became assistant to Robert H. Sayre, the Chief Engineer. Feeling that there was a better future in mining than in civil engineering, he left the service of the Lehigh Valley R. R. Co., and again became associated with Mr. Stearns. In 1881, he accepted the superintendency of an iron company in New Jersey. He next went to Virginia as Superintendent and Engineer of the Southwest Virginia Improvement Co., the pioneer enterprise in the now famous Pocahontas region of the Norfolk & Western railroad, where he built the town of Pocahontas and developed the well-known Pocahontas coal-field. In 1884, he returned from Virginia and settled at Snow Shoe, Pa., where he had the management of the Lehigh Valley Coal Co.'s bituminous-coal department. On Feb. 1, 1888, he took charge, as Chief Engineer, of all the mining operations of the Lehigh Valley Coal Co., succeeding Mr. Fred Mercur, who had died in January of that year. He was shortly after made General Superintendent, and later General Manager. He resigned this position in June, 1901, to become President of the Webster Coal & Coke Co., and on Feb. 1, 1903, was made President of the Pennsylvania Coal & Coke Co., and also President and Director in the various subsidiary companies controlled by that company. He was elected President of the Lehigh Coal & Navigation Co., Feb. 26, 1907. This company, in addition to its extensive coal-holdings, owns and leases to the Central railroad of New Jersey, the Lehigh & Susquehanna railroad from Easton, Pa., to Scranton, Pa. It also owns and operates the Lehigh and Delaware canals from Mauch Chunk to Philadelphia Harbor, and the Lehigh & New England railroad from the company's mines at Lansford, Pa., to the Poughkeepsie Bridge, New York. During the five years of Mr. Lathrop's administration the annual production was increased 58 per cent.; the average yield per mine-car 33 per cent., and the yield of prepared coal per mine-car 40 per cent. A briquetting-plant for utilizing the fine coal formerly wasted was built and is now in successful operation. A mine-fire, started in 1859, endangered a large part of the company's coal, and large sums of money had been expended continuously for 50 years in an effort to control it. But this attempt had been unsuccessful, and the fire threatened seriously to enter the main Panther Creek basin. Mr. Lathrop, taking personal supervision at this crisis, decided that a barrier must be placed across the Summit Hill basin, to cut off the fire effectually and save the coal in the main basin. It was a difficult and dangerous undertaking, and its successful accomplishment required five months of continuous work by day and night. This work is considered one of the great feats of mining engineering in the anthracite coal-fields. A full account of it was given in a paper read by Mr. Lathrop at the meeting of the American Institute of Mining Engineers, held at Glen Summit, Pa., in June, 1911.

He also extended the Lehigh & New England railroad from Danielsville, Pa., a distance of 32 miles, and by joining it with the local mining railroad of the company at Lansford gained for the company an independent rail-outlet for its coal over practically all the large railroad lines in the East.

In order to utilize the cheap grades of the small sizes of anthracite which have been for years wasted, he organized the Lehigh Navigation Electric Co., a subsidiary company, to transmit power through the industrial regions of the Lehigh Valley to tidewater. This plant, designed for 100,000 kw., will be in operation in 1913.

Mr. Lathrop was also largely interested in bituminous coal-mines in West Virginia and Kentucky. He was President of the Board of Trustees of Lehigh University, a Director of the Peoples Bank of Wilkes-Barre, of the Fourth National Bank and the Guarantee & Trust Co. of Philadelphia, a member of the Union League, University, and Art Clubs of Philadelphia, the Westmoreland and Country Clubs of Wilkes-Barre, The Pennsylvania Society of the Sons of the Revolution, and of the Wyoming Historical and Geological Society.

He became in 1883 a member of the Institute, of which at the time of his death, which took place April 12, 1912, he was a Vice-President. His cordial participation in the sessions of the Canal Zone meeting of 1910 endeared him personally to many of his fellow-members.

Mr. Lathrop was endowed with great natural gifts, and a charming personality, true to his friendships, yet loyal to duty. He was indeed one of Nature's noblemen. What more can be said?

I. A. S.

SUBJECT TO REVISION.

[TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS.]

Notes on Ruff's Carbon-Iron Equilibrium Diagram.

BY HENRY M. HOWE, NEW YORK, N. Y.*

(Cleveland Meeting, October, 1912.)

Manuscript received Aug. 20, 1912.

PROFESSOR RUFF's most illuminating paper[1] describing his extremely valuable investigation of the carbon-iron equilibrium diagram assigns definite temperatures to certain very important lines in the diagram, viz.: the eutectic temperature *EB* (Fig. 1) and the "near-eutectic" temperature *E′B′*, but without indicating clearly enough the reasoning which warrants these assignments. These temperatures are of such primary importance that it is well to indicate the apparent gaps in the reasoning, in the hope that they may be filled in.

In his diagram, Fig. 1, the point *X* and the lines *B′D′HI* represent the experimental results reached by Ruff and Goecke;[2] *AB*, *AEB*, and *SE* represent the liquidus, the solidus, and the solubility of the carbon in solid austenite in the absence of graphite according to Gutowsky,[3] *AB* being also in accordance with Spiecker's data;[4] *S′E′* the solubility of carbon in solid austenite in the presence of graphite according to Ruer and Iljin;[5] *GS* the line *A3* after Goerens and Meyer;[6] and *EBC* the true eutectic line as inferred by Prof. Ruff from Gutowsky's data. *B′D′* is the carbon content found in cast-iron which had apparently saturated itself with carbon in the presence of the graphite of the enclosing graphite crucibles in which the determinations were made. From its straightness and its sharp inflection near 6.67 per cent. of carbon, the carbon content of cementite, Fe_3C, he infers that the carbon exists in the molten iron almost solely in the form of cementite in this range of temperature. In a molten alloy above *B'D'* the dissolved cementite is stable. If it passes below *B'D'* all cementite then in excess of *B'D'* is thereby made metastable and tends to precipitate, and that which so precipitates graphitizes promptly. *BD*, not determined experimentally, represents the true solubility of cementite in molten iron, a line on reaching which the precipitation of cementite accelerates.

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* Professor of Metallurgy in Columbia University.

[1] *Metallurgie*, vol. viii., pp. 456 to 464 and 497 to 508 (1911).

[2] *Idem*, vol. viii., pp. 417 to 421 (1911).

[3] *Idem*, vol. vi., pp. 731 to 743 (1909).

[4] *Idem*, vol. vi., p. 524 (Wüst) (1909).

[5] *Idem*, vol. viii., pp. 97 to 101 (1911).

[6] *Idem*, vol. vii., pp. 307 to 312 (1910).

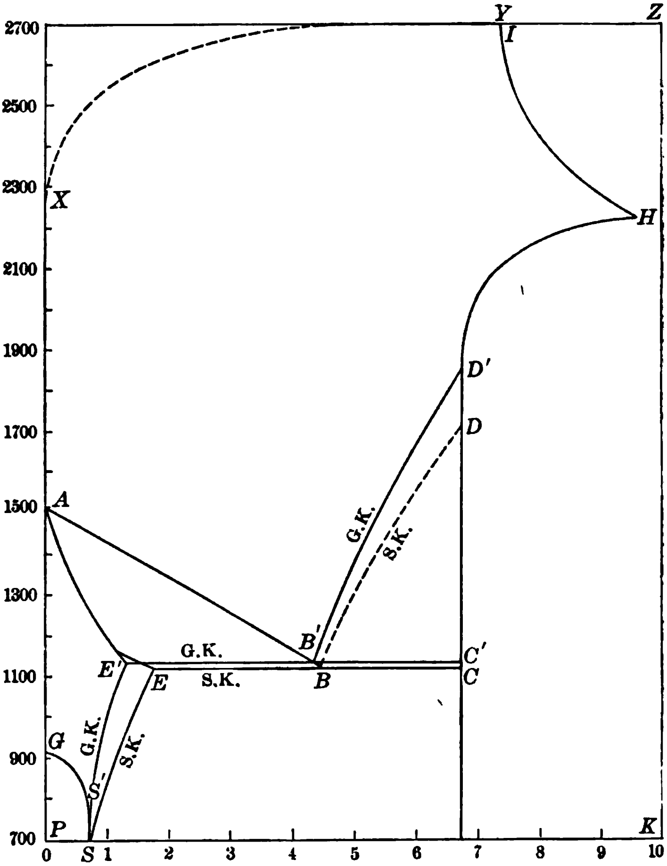

FIG. 1.—RUFF'S CARBON-IRON EQUILIBRIUM DIAGRAM. THE *GK* LINES REPRESENT EQUILIBRIUM WITH GRAPHITE AS BODENKOERPER, THE *SK* LINES WITH CEMENTITE AS BODENKOERPER.

The later results of Wittorff,[7] not yet readily accessible, are at variance in many respects with Ruff's for high carbon contents.

Let us now review the conditions underlying the matter which I discuss, in the form of 20 propositions in which I attempt to give the premises from which Prof. Ruff reaches his conclusions. My general contention is that, granting these propositions, the steps in Prof. Ruff's reasoning ought to be stated more fully in order to carry conviction. Indeed, if we make in proposition (15) the modification explained in (32), and this seems a permissible one, then the data seem explicable in the simpler way set forth in (28) to (31). Further, the Gutowsky data on which Prof. Ruff bases his calculations seem insufficient in view of the capriciousness of the phenomena and the difficulty of reconciling some of them with our natural conceptions. My purpose thus is to bring the matter up for discussion, and only as a basis for discussion do I offer my tentative explanation. The matter enclosed in parentheses in the text of these twenty propositions represents ideas of my own for which Professor Ruff is not responsible.

General Outline.—(1) to (21), pp. 1183 to 1193, general outline of Ruff's premises.

(22), p. 1194, general meaning of his diagram.

(23) to (26), pp. 1194 to 1199, Ruff on the eutectic temperature.

(26A) to (35), pp. 1199 to 1207, other positions of the eutectic temperature.

(36) p. 1208. Capriciousness of the rate of graphitization and of the arrest and recalescence temperatures.

(37), p. 1213. Graphitization goes on at temperatures below those admitted by Prof. Ruff.

(38) p. 1216. How dominant is the bodenkoerper?

(39) p. 1218. Conclusion.

(1) Carbon when dissolved either in the molten iron between say 1,800° and 1,135°, or in the austenitic solid iron above the bottom of the transformation range, A1 (say 725°), exists almost solely as tri-ferrous carbide, Fe_3C, called cementite. (P. 460, ¶ 2.) These and the following page numbers refer to Prof.

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[7] *Journal of the Russian Physico-Chemical Society*, vol. xliii., pp. 1613 to 1690 (1911); through *Chemisches Zentralblatt*, part 1, pp. 1091 to 1094 (1912); *Journal of the Society of Chemical Industry*, vol. xxxi., p. 388 (1912).

Ruff's paper, *Metallurgie*, vol. viii., 1911. The section numbers in parentheses refer to the present paper.

(2) Between these temperatures, say 725° and 1,800°, the quantity of carbon which exists in the elemental state in solution in the iron is so small that the true graphite-iron diagram plays only an indirect part. (P. 458.) (Some might be inclined to regard it as hardly more than a metaphysical abstraction.) The diagram which iron in solidifying actually follows is the cementite-austenite diagram, though this too is in a sense a double diagram, one set of lines representing equilibrium when graphite is the "bodenkoerper," and the other set representing true solubility, *e. g.*, when cementite is the bodenkoerper, a term the meaning of which is set forth in Appendix II.

(3) The line *B′D′*, experimentally determined by Ruff and Goecke,[8] represents the quantity of carbon as cementite which the molten iron holds when in equilibrium with solid graphite as bodenkoerper (pp. 458, 459, 460) and the line *S′E′* (p. 456), adopted from Ruer and Iljin,[9] represents that which the solid austenite holds when in equilibrium with graphite as bodenkoerper, as in gray cast-iron. The line *BD* (p. 500) represents the true solubility of cementite in molten iron, and *SE* (p. 456) its true solubility in solid austenite, *i.e.*, the quantity which the molten iron and the solid solution or austenite hold in equilibrium with solid cementite. This line is important because on reaching it the precipitation of cementite accelerates; and its importance does not rest on any assumption as to the ability of free cementite to resist graphitization at these very high temperatures. (In (38) facts are given which are not at first reconciled easily with this idea that the nature of the bodenkoerper is of the first importance in determining whether the dissolved carbon content shall follow *SE* or *S′E′*.)

(4) From (2) it follows that, when the dissolved carbon content or concentration of the molten is that represented by *B* (Figs. 1 and 2), or in undercooling by a point on BB_4 (Figs. 2 and 3), the dissolved carbon content of the solid austenite then depositing from that molten should be represented by the point horizontally opposite on *SE;* and that when the dissolved carbon content of the molten is represented by *B′*, or in undercooling by

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[8] *Idem*, vol. viii., pp. 417 to 421 (1911).

[9] *Idem*, vol. viii., pp. 97 to 101 (1911).

a point on $B'Q$, the dissolved carbon content of the solid austenite then depositing should be represented by the point horizontally opposite on $S'E'$. Conversely, the fact that the dissolved carbon content of the austenite depositing is represented by a point on SE or $S'E'$, as the case may be, implies that the dissolved carbon content of the molten from which that austenite is depositing should be represented by the point horizontally opposite on BB_4 or B'. (The conception that in undercooling the course of the depositing layers is along $E'E$ need not here disturb us. For instance if, in undercooling, the

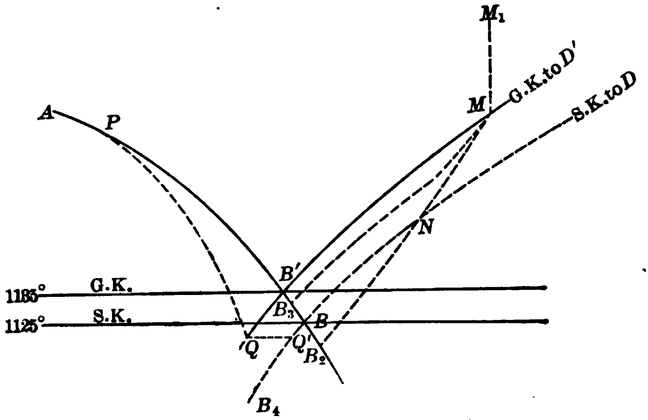

FIG. 2.—THE COURSE OF THE MOLTEN DURING SOLIDIFICATION, APPROXIMATELY AFTER RUFF.

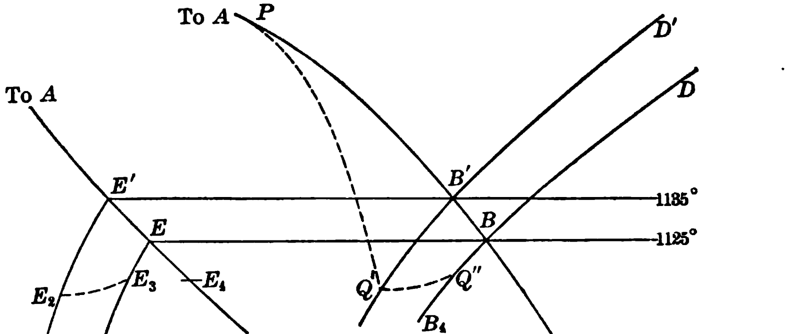

FIG. 3.—THE COURSE OF THE MOLTEN AND OF THE SOLID DURING SOLIDIFICATION, APPROXIMATELY AFTER RUFF.

course of the molten down to the trough of the recalescence is $PQ'Q''$ (Fig. 3), and if the course of the solidifying layers is correspondingly $E'EE_4$, then for instance at E_4, when the molten reaches SK (see (5)) at Q'', the dissolved carbon content of the austenite is E_3, the point horizontally opposite on SE, and the difference, E_4E_3, is represented by free cementite, which graphitizes rapidly at this temperature. And in general while the molten is crossing from Q' on GK to Q'' on SK, the austenite of the depositing layers is crossing from E_2 to E_3 on SE.)

(5) GK and SK. For brevity $B'D'$ and $S'E'$, and $B'Q'$ the prolongation of $B'D'$ in undercooling, and the horizontal $E'B'$ which connects the points at which they end in equilibrium he calls GK; and BD and SE, and BB_4 the prolongation of BD in undercooling, and the horizontal true eutectic line EB which connects E and B in equilibrium he calls SK (*Gleichgewichtskonzentration* and *Saettigungskonzentration* respectively, pp. 458 and 463).

Nomenclature.—Ruff and some others use the unprimed letters SE for the part of SK which lies below the freezing range, but the primed letters $B'D'$ for that part of it which includes the freezing range and the molten, using $S'E'$ for GK in the solid and BD for GK in the molten. But because it is simpler to represent the whole of one line by means of unprimed and the whole of the other line by primed letters, and because the most firmly fixed and also the earliest usage is that which associates unprimed SE with SK, I represent SK for the solid and for the molten respectively by SE and BD, and GK by $S'E'$ and $B'D'$.

In the same way the GK or "near-eutectoid" temperature at which $S'E'$ cuts GOS may be called $A'1$.

(6) All carbon dissolved as cementite up to GK is strictly stable (p. 463).

(7) Carbon dissolved as cementite in excess of GK and up to SK is metastable, and tends to precipitate and graphitize, $Fe_3C = 3Fe + Gr$ (p. 463). (It is simplest to assume that the graphitization is confined to cementite which actually precipitates. See p. 462, ¶ 4, lines 12 to 14).

(8) EB, the horizontal which connects the points at which SE and BD end in equilibrium, is the true metastable or cementite-austenite eutectic temperature (p. 463 and p. 498, line 6).

(9) *E'B'*, the horizontal which connects the points at which *S'E'* and *B'D'* end when in equilibrium, is no true eutectic line, but like the lines which it connects merely the temperature on cooling past which cementite changes progressively from being dissolved in perfect stability to being dissolved in metastability, tending to precipitate out and if so precipitated to graphitize. (For brevity it might be called a "near-eutectic" line) (p. 462).

(10) The carbon content of *E'*, *i. e.*, the solubility of carbon in solid austenite when graphite is bodenkoerper at temperature *E'B'*, is taken from Ruer and Iljin as 1.3 per cent.[10] (p. 456).

(11) The carbon content of *E*, *i. e.*, the true solubility of cementite in solid austenite at temperature *EB*, cementite being bodenkoerper, is taken from Gutowsky as 1.7 per cent.[11] (p. 456).

(12) Carbon dissolved in excess of *SK* precipitates promptly as cementite (p. 498). (I find no direct enunciation of this, but the reasoning evidently assumes this.)

Thus, there are four conditions of cementite: (A) stable up to *GK;* (B) metastable and tending to precipitate, *GK* to *SK;* (C) in excess of *SK* (labile?) precipitating promptly; and (D) that actually precipitated, whether from (B) or (C).

(13) Cementite actually precipitated graphitizes with great rapidity at temperatures above 1,135°. Professor Ruff's language leaves on one the impression that this graphitization is well nigh instantaneous. But Hanemann's results[12] show that, rapid as it is, it can yet be restrained by a rapid enough cooling. Wittorff's language also implies that the carbide which here precipitates can be retained ungraphitized, though as to the nature of this carbide he and Ruff disagree.[13]

(14) Beginning at 1,135°, with further fall of temperature graphitization is retarded very greatly, so that, according to Prof. Ruff's calculations, whereas at 1,135° the time occupied by the graphitization of 50 per cent. of the cementite precipitated in ex-

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[10] *Metallurgie*, vol. viii., p. 99 (1911).

[11] *Idem*, vol. vi., p. 739 (1909.)

[12] *Stahl und Eisen*, vol. xxxi., pp. 333–336 (1911).

[13] *Journal of the Russian Physico-Chemical Society*, vol. xliii., pp. 1613 to 1690 (1911); through *Chemisches Zentralblatt*, part 1, pp. 1091 to 1094 (1912); *Journal of the Society of Chemical Industry*, vol. xxxi., p. 388 (1912).

cess of *GK* is 2 sec., this time doubles with every 10° fall of temperature, becoming 32 sec. at 1,085° (or 1,095°) (pp. 502 and 507). These numbers refer to graphitization during solidification, when the cementite crystals are still bathed with molten iron. The rate of graphitization within solid iron Prof. Ruff seems to regard as negligibly slow. (If, as seems reasonable, we regard the graphitization of cementite as taking place chiefly through a process of solution, diffusion, and re-precipitation, then because diffusion is much more rapid in a molten than in a solid solution, it is but natural that graphitization should be much faster in molten than in solid metal. Prof. Ruff's numbers evidently apply only in the nearly complete absence of silicon, an element which hastens graphitization very greatly. The data in the experiments of Heyn and Bauer[14] indicate that, in their irons containing 4.16, 1.58, and 0.63 per cent. of silicon, the rate of graphitization at temperatures below 1,135° is much greater than that which Prof. Ruff calculates. West's experiments too point to rather rapid graphitization far below the freezing-range in the presence of silicon. See 37).

(15) *Pro-Eutectic Solidification of Hyper-Eutectic Alloys* (pp. 463, 464).—The cementite in an alloy of carbon content *M* (Fig. 2) and at temperature M_1 is wholly stable (6), and remains stable till in cooling the alloy starts to cross *B′D′* at *M*. The start to cool below *B′D′* makes so much of the cementite as is in excess of *B′D′* metastable, and because of this metastability part of it precipitates and at once graphitizes (7). But because the precipitation is not complete, some of the cementite thus metastabilized remains unprecipitated, and hence the molten (as represented by its temperature and dissolved carbon content) passes somewhat to the right of *B′D′*, and follows a path which varies with the rate of cooling, *i.e.*, there is a certain degree of undercooling below *B′D′*. If the removal of heat is rapid, the dissolved carbon content of the molten may fall to *BD*, as for instance by the path *MN*. But because on starting to pass below *BD*, the curve of true solubility, the precipitation is extremely prompt, the molten is not likely actually thus to pass below *BD*, but rather to slide down along *BD*

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[14] *Stahl und Eisen*, vol. xxvii., pp. 1565 to 1571 and 1621 to 1625 (1907).

from N to B. In the case of an extremely rapid cooling there may indeed be undercooling even below BD, so that the molten travels along MNB_2. With a moderate (*mittleren*) rate of cooling the molten follows a path like MB_3, *i.e.* with a moderate undercooling below $B'D'$ but not reaching BD. (The line which he actually sketches (Fig. 2) for cooling of moderate rapidity differs slightly from MB_3B, reaching B without previously reaching the line $AB'B$; but his language (p. 464, lines 10 to 12) implies that MB_3 as I give it is that which he actually intended.)

(16) *Near-Eutectic and Eutectic Solidification of Hyper-Eutectic Alloys* (p. 497).—But whether the course of the molten is like MNB_2 or MNB or MB_3, as soon as it reaches the austenite-liquidus AB or its continuation $B'BB_2$ the precipitation of austenite sets in, and the latent heat of this precipitation is added to the still evolving heat of precipitation and graphitization of the cementite metastabilized by the passage to below $B'D'$ (pp. 464 and 497). At the same time the heat supply from the precipitation and graphitization of cementite is greatly increased (p. 497). For the precipitation of the carbon-poor austenite from the carbon-rich molten tends to raise the carbon content of that molten rapidly further to the right of $B'D'$; and the farther it passes to the right of $B'D'$ the greater the metastability and the more rapid the precipitation and graphitization of cementite.

These heat evolutions combined suffice, even if the external conditions are such as to drain heat away relatively fast, to raise the temperature of the molten again to 1,135°, so that it slides up BB' to B', but of course not beyond B', because the start to cross $B'D'$ would arrest the metastabilizing of cementite and the heat evolution resulting from the precipitation and graphitization of that cementite, and without heat from this source a rise of temperature would be impossible.

As the quantity of residual molten decreases, the heat of precipitation from it and the heat of graphitization cease to be able to hold the temperature as high as B'. As the temperature therefore sinks, the course of the molten is naturally along $B'B$, neither falling below $B'B$ because this would further accelerate the precipitation of austenite and thus check the cooling, nor above it for that would arrest completely the precipitation of austenite, cut off this important source of heat, and drag the temperature instantly back to $B'B$.

This cooling along $B'B$ continues till the molten reaches B, the true eutectic temperature and dissolved carbon content for the molten. This arrival at SK, this substitution of the true solubility content for mere metastability because in excess of GK, so accelerates the precipitation of cementite (12) that the temperature and dissolved carbon content of the molten now remain constant at B till solidification completes itself eutectic-wise.

The mechanism of the increase in the dissolved carbon content of the molten from GK at B' to SK at B has two features which we may notice. (A) Because the precipitation of the cementite metastabilized by being in excess of GK is incomplete, some of it accumulates in solution in the molten, raising the dissolved carbon content of that molten from GK for the existing temperature towards SK. (B) Because the graphitization of cementite actually precipitated, though rápid, is not instantaneous, some of it (*ein letzter Rest*) accumulates ungraphitized and suspended in the molten, where, acting as bodenkoerper, it in part re-dissolves in the attempt to raise the dissolved carbon content of the molten. in the immediate sphere of its influence to SK for the existing temperature. Thus arises a molten enriched beyond GK, and hence normally solidifying at a temperature below B' (p. 497).

Of the eutectic cementite, part diffuses back to enrich towards SK (SE) the early austenite layers deposited when the molten was on $AB'B$ but between B' and B, so that those layers had a dissolved carbon content less than SE; and part graphitizes during the remainder of the eutectic freezing (and thereafter) (p. 497).

Here notice the distinction between the "near-eutectic" co-precipitation of austenite and cementite at points on $B'B$ above B, and the true eutectic precipitation at B. Above B the precipitation of cementite has only such moderate promptness as is due to its metastability because in excess of GK; at B it has the promptness due to the molten's being on the true solubility line, SK.

To sum this up, the course of the molten is $M_1MB_3B'B$.

(17) *Pro-Eutectic Solidification of Hypo-Eutectic Alloys, Under-*

cooling is Marked.—The approach of the molten part of the hypoeutectic irons to 1,135° in cooling, occurring theoretically along AB', *i.e.*, without liberation of cementite, they lack the heat evolved due to the prompt and continuous graphitization of cementite which occurs in hyper-eutectic irons as they cool along $B'D'$, towards AB', 1,135°. Lacking this heat supply their undercooling is more pronounced, as, for instance, along the line PQ, Fig. 2 (p. 497).

It is indeed so marked that the temperature at which the molten reaches GK may be as low as 1,125° or even lower (p. 498, top). Till $B'D'$ is reached the solidification is of course of austenite only. On reaching and starting to pass $B'D'$ the cementite in the molten thereby brought into metastability, *i.e.*, excess over GK, starts to precipitate and graphitize. This at first causes only a retardation of the cooling, but as GK is farther exceeded the precipitation and graphitization of the cementite thus made metastable in consequence accelerate. Because of this acceleration of graphitization this retardation of cooling is quickly followed by an arrest of cooling (sojourn at constant temperature), but hardly by a material (*wesentlich*) rise of temperature (pp. 497–98), because the rapidity of graphitization of this metastable dissolved cementite can hardly be rapid enough to cause one, (in view of the presence of the already precipitated austenite, in a position favorable for absorbing its share of all heat evolved).

The further out-crystallizing of austenite enriches the molten in dissolved cementite faster than the lagging precipitation of the metastable cementite can impoverish it, so that the course of the molten is to the right along QQ', and at last it reaches SK at Q'. (As in the case of hyper-eutectic alloys, so it is here. The arrival at Q inducing the precipitation of some cementite, there is "near-eutectic" precipitation in the sense that both austenite and cementite precipitate, though the precipitation of cementite, being only that due to its metastability because in excess over GK, is less prompt than at SK.)

(18) *The Eutectic Solidification of Hypo-Eutectic Alloys* (p. 498).—The arrival at SK, whether at its equilibrium temperature B, or because of undercooling at some point like Q' on its prolongation BB_4, substitutes the prompt precipitation of cementite for the lagging precipitation which has thus far been

induced by the mere metastability of so much of the dissolved cementite as is in excess of *GK*. Thus the co-precipitation of austenite and cementite which has been going on now continues, but the acceleration of the precipitation of cementite now induced brings that co-precipitation to the eutectic ratio of austenite to cementite, so that now for the first time there is strict eutectic co-precipitation.

(19) *Recalescence Begins only on the Arrival at SK* (p. 498). —This acceleration of the precipitation and consequent graphitization of cementite so accelerates the heat evolution that now for the first time can a material rise of temperature occur (p. 498, lines 11 to 18). But this rise may not reach the true eutectic temperature, and is the less likely to the greater the initial hypo-eutecticness, *i.e.*, the farther the alloy falls short of the eutectic ratio of 4.3 per cent. of carbon. In short, the molten will now travel along *SK* from *Q′* towards or perhaps even to *B*.

To sum up, the course of the molten in hypo-eutectic alloys is *APQQ′* and up *Q′B* to or towards *B*, Fig. 2.

(20) *Contrast Between Hyper- and Hypo-Eutectic Alloys.*—(A) A recalescence, in the strict sense of an actual rise of temperature, can occur in hyper-eutectic alloys before the dissolved carbon content of the molten reaches *SK*, *e.g.*, when it reaches *APB′B* at B_3; but in hypo-eutectic alloys no material rise can occur before the molten thus reaches *SK*. (B) The undercooling is more marked in hypo- than in hyper-eutectic alloys.

In order to understand this we must distinguish during the solidification between the calorific molten and the inert already solid layers. Whatever heat is generated in the molten must, in order to cause an actual recalescence, be spread over both molten and solid. In the case of hyper-eutectic alloys the quantity of solid, at any given stage in the pre-eutectic solidification, is extremely small, consisting of little more than the small quantity of graphite generated by the decomposition of the cementite thus far precipitated. Thus when a hyper-eutectic alloy in cooling along MB_3 reaches *APB* there is but an insignificant quantity of solid graphite, and the mass is almost solely calorific molten. Hence the co-precipitation of austenite and cementite which here begins suffices to raise the temperature of the whole, even though the precipitation of the cementite is the relatively sluggish one caused by its metastability because in excess of *GK*.

But when a hypo-eutectic alloy in sliding down *APQ* reaches *Q*, there is now present the large quantity of solid carbon-poor austenite precipitated in the cooling along *APQ*. The thermal capacity of this inert solid austenite is so great that the ensuing co-precipitation of austenite and cementite does not suffice to raise the temperature till the precipitation of cementite reaches its full speed on the arrival of the molten at *SK*.

(Two co-operating ways suggest themselves in which the presence of an abundant primary or pro-eutectic substance may increase eutectic undercooling: (A) By mechanically breaking up the residual molten eutectic and isolating its several microscopic particles from each other, it impedes the spread of the influence of the nucleus action of those molecules of the eutectic which are the most prompt to solidify. (B) As for given conditions the undercooling increases, so do the eutectic solidification and the consequent heat evolution hasten. An arrest and a recalescence mean that this acceleration at length suffices first to balance and then to overcome the escape of heat to the environment. Because of the capacity of an abundant pro-eutectic substance, especially if it has great thermal conductivity, both for absorbing heat and for transmitting it to the environment, its presence implies that the arrival at this balance and at this overcoming requires a more rapid heat evolution, and consequently a greater undercooling.

In a word, in a hyper-eutectic alloy the primary solid is extremely scanty because it is almost pure carbon, whereas in a hypo-eutectic one it is extremely abundant because it is carbon-poor austenite. Hence the former may recalesce without the full speed *SK* generation of cementite; the latter cannot).

With the exception of the words enclosed in parantheses the 20 propositions which I have just given are my understanding of Professor Ruff's own interpretation of the course of solidification. Much of it, for instance (8), I do not find distinctly enunciated, but read it rather hazardously between the lines.

(21) His forecast of the course of the molten during the solidification of hypo-eutectic alloys may be summed up thus: (A) The temperature begins to slacken its fall on reaching *GK*; (B) then stays nearly constant, with perhaps a slight rise;

till (C) on reaching the prolongation of *SK* (BB_4) it starts to rise materially; but (D) it may or may not reach the eutectic temperature *EB* (p. 498).

(22) *General Meaning of Professor Ruff's Diagram*.—Looking at this diagram in a general way, we cannot regard it as a "double diagram" in the sense in which these words have often been used. That is to say, it has no lines representing the solubility of graphite in solid and molten iron respectively, or, in other words, the lines at which in cooling the precipitation of graphite begins. *S'E'* and *B'D'* do not represent graphite solubility. They represent solely the equilibrium content of cementite in solid and molten iron respectively in the presence of graphite as bodenkoerper, as distinguished from the solubility proper of cementite, *i. e.*, its equilibrium content in the presence of cementite as bodenkoerper. And, because *SE* and *BD* represent true solubility, it is the horizontal *EB* connecting their ends that is the eutectic line. *E'B'*, then, is no true eutectic line, though it is a line at which in equilibrium there is an acceleration of solidification, because on cooling to it both components of the liquid solidify, whereas in the cooling before it is reached, in hyper-eutectic irons it is only their cementite, and in hypo-eutectic ones it is only their austenite, that solidifies. Yet if the true eutectic line is later reached, then solidification is further accelerated. For the precipitation of cementite on reaching its true solubility curve *BD* is more rapid than the precipitation due to its metastability because in excess of *B'D'*.

So too in undercooling. For instance, though in the case of a hypo-eutectic iron there is a certain acceleration of solidification and of heat evolution when the molten, in sliding down *PQ*, reaches *GK* at *Q*, there is a further acceleration when it reaches the prolongation of *SK* at *Q'*. And, in the case of a hyper-eutectic iron, though there is an acceleration of solidification and of heat evolution when the molten undercooled below *GK*, sliding along MB_3 reaches the austenite liquidus at B_3, there is a farther acceleration when it, in turn, on sliding along AB_3B at last reaches *SK* at *B*.

(23) *Ruff on the Temperature of the Eutectic Line EB*.—By applying the foregoing considerations to the cooling curves of Gutowsky, Fig. 4 and Table I., Professor Ruff infers that the

eutectic temperature is I. below 1,130°; II. above 1,123°; and III. only slightly above 1,123°, so as to be taken reasonably at 1,125°. Proposition II. seems to be well founded; but the gaps in the reasoning by which he infers I. and III. are not readily filled. Let us take these up briefly in the order II., III, and I.

(24) II. *The Eutectic Temperature EB is at or above* 1,123°.—The reasoning by means of which this inference is reached is only hinted at; it may be constructed somewhat as follows:

(A) Gutowsky's alloy No. 4, with most of its arrest at 1,123°, contained 2.184 per cent. of combined carbon. (B) Of this, the largest quantity which the solid austenite could contain in solution is 1.7 per cent., see (11), so that at least 2.184 — 1.7 = 0.484 per cent. must have existed as free cementite during solidification, because, though cementite may be broken up by graphitization, cementite cannot increase in quantity after solidification. (C) The fall of temperature at the end of the arrest was so sharp as to indicate that no important heat evolution occurred after the arrest, and hence this 0.484 per cent. of free cementite must have been generated during the arrest. (D) But the graphitization of free cemetite at a temperature as high as 1,123° is still extremely rapid from Ruff's calculations, say 50 per cent. in 4 sec. Hence the persistence ungraphitized of this 0.484 per cent. of free cementite implies that it must have been generated very near the end of the arrest, and hence very rapidly. (E) But it is only when the carbon content of the molten reaches *SK* (B_4BD, Fig. 2) that free cementite is generated rapidly; hence, when this 0.484 per cent. of free cementite was generated, *i. e.*, during the arrest at 1,123°, the molten

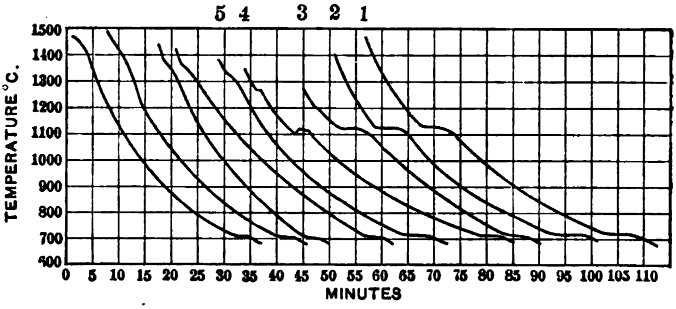

Fig. 4.—Gutowsky's Cooling Curves.

TABLE I.—*Influence of Total Carbon Content on the Condition of the Carbon, and on the Temperature of the Eutectic Solidification.*

(From the data of Gutowsky, Heyn and Bauer, Carpenter and Keeling, and Wüst.)

Reference No.[a]	Authority.	Silicon.	Carbon Content.						Solidification.		Length of Arrest from Time of Reaching Trough to Again Sinking to Temperature of that Trough.	Rate of Pro-Eutectic Cooling from 1,180° to 1,140°.
			Total.	Graphite.	Combined.	Proportion of Total Carbon Graphitized.	Combined in Excess of 1.3 (E').	Combined in Excess of 1.7 (E).	Trough of Recalescence.	Crest of Recalescence.		
		Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	°C.	°C.	Seconds.	Seconds.
1	Gutowsky, First Series.	0.091	4.940	3.910	1.030	79.2	(—0.270)	(—0.670)	1,130	1,138	225	105
2		0.090	4.626	3.413	1.213	73.8	(—0.087)	(—0.487)	1,130	1,134	260	90
3		0.064	3.677	2.263	1.414	61.5	0.114	(—0.286)	1,123	1,130	250	105
4		0.060	2.795	0.611	2.184	21.8	0.884	0.484	1,100	1,123	110	75
5		0.048	2.165	0.282	1.883	13.0	0.583	0.183	1,093	1,093	25	90
6		0.045	1.762	......	1.762	......	0.462	0.062	....	....	....	....
7	Gutowsky, Sec. Series	....	3.801	....	....	....	....	....	(1,134)	1,134	110	30
8		....	3.596	....	....	....	....	....	(1,130)	1,130	92	40
9		....	2.742	....	....	....	....	....	(1,125)	1,125	52	30
10		....	1.913	....	....	....	....	....	1,062	1,066	13	25
									Temperature of Arrest[b] °C			
11	Heyn and Bauer.[b]	1.58	3.28	2.03	1.25	61.9	(—0.05)	(—0.45)	1,135		270	180 about
12		0.63	3.19	0.25	2.94	78.4	(—1.05)	(—1.45)	1,105		205	180 about
13		4.16	3.12	2.68	0.44	85.7	(—0.86)	(—1.26)	1,133		220	140 about
14	Capenter and Keeling.	....	3.98	....	....	....	....	....	1,138		240	240 about
15		....	3.87	2.14	1.73	55.3	0.43	0.03	1,144		....	
16		....	3.57	1.38	2.13	39.3	0.83	0.43	1,136		....	
17		....	3.42	1.80	1.62	52.6	0.32	(—0.08)	1,114		....	
18		....	3.29	0.14	3.15	4.3	1.85	1.45	1,141		....	
19		....	3.03	Nil	3.03	Nil	1.73	1.33	1,139		....	
20		....	2.12	Nil	2.12	Nil	0.82	0.42	1,110		40	240 about
21	Wüst.[b]	0.012	4.66	3.22	1.44	69.1	(—0.26)	0.14	1,130		270	50
22		0.039	4.04	1.75	2.29	43.3	0.59	0.99	1,122		210	40
23		0.106	3.94	2.28	1.66	57.8	(—0.04)	0.36	1,149		290	50
24		0.013	3.79	2.24	1.55	59.1	(—0.15)	0.25	1,141		850	60
25		0.009	3.76	2.33	1.43	62.0	(—0.27)	0.13	1,138		310	200 about
26		0.015	3.29	0.05	3.24	1.5	1.54	1.94	1,112		130	100 about
27		0.008	3.02	0.37	2.65	12.2	0.95	1.35	1,135		115	80
28		0.006	2.94	0.04	2.95	1.3	1.25	1.65	1,132		80	....

[a] References: **1 to 10.** Gutowsky, *Metallurgie*, vol. vi., pp. 732–733, and 738 (1909).

11 to 13. Heyn and Bauer, *Stahl und Eisen*, vol. xxvii., pp. 1566, 1569, and 1570 (1907).

14 to 20. Carpenter and Keeling, *Journal of the Iron and Steel Institute*, 1904, No. I, pp. 229–231. (The silicon in these alloys was very probably below 0.20 per cent.).

21 to 28. Wüst, *Metallurgie*, vol. iii., pp. 1–13 (1906).

[b] Nos 11 to 13 and 21 to 28 are based on inverse rate curves. The temperatures here given are those of the greatest retardation. These curves do not show whether there was either an arrest or a recalescence, strictly speaking.

In the Carpenter and Keeling cases Nos. 14 and 20 the cooling curves show no recalescence as distinguished from a retardation. Those for 15 to 19 are not given, and here we are left in doubt as to whether there was a recalescence.

must have been on or below B_4BD. (F) Moreover, it must have been at or below $AB'B$, because a heat evolution rapid enough to cause an arrest could occur only in case the latent heat of solidification of austenite is then evolving; and austenite can solidify only at or below $AB'B$. In short, from (E) the molten must have been on or below B_4BD, and from (F) it must have been on or below $AB'B$, and hence (G) it must have been on or below B_4BD at or below B. Because its temperature was 1,123°, B must be at or above 1,123°, Q. E. D.

(25) III. *The Eutectic Temperature EB must be only slightly above* 1,123°.—The reasons for this inference are hinted at so vaguely that the gaps cannot be filled with confidence. One conjectures that the reasoning is somewhat as follows. Whereas the difficulty is to explain how so much cementite remains ungraphitized in spite of the rapidity of graphitization at the high arrest temperature, 1,123°, at which it was generated; and whereas it is only the undissolved or free cementite that is subject to graphitization; and whereas, in so relatively rapid a cooling as Gutowsky's, the cementite precipitated after the arrest and during the subsequent cooling would graphitize but slightly, so that it is the persistence of the cementite free at the time of the arrest at 1,123° that needs explanation; therefore the difficulty is reduced to a minimum by assuming at the time of the arrest the largest quantity of dissolved and the smallest quantity of free cementite, *i.e.*, by assuming that the dissolved cementite represents 1.7 per cent. of carbon, the maximum solubility, or at least that it represents a quantity only very slightly less. If the carbon content represented by the dissolved cementite were 1.7 per cent., then because that is the carbon content of austenite deposited at eutectic temperature, represented by point E on SE, it would follow that the temperature at which it was deposited was the eutectic temperature. If the dissolved cementite were only very slightly below 1.7 per cent., then because the temperature on SE at which such austenite is deposited is only very slightly below the eutectic temperature, it follows that the observed temperature of deposit and arrest, 1,123°, is only very slightly below the eutectic temperature.

(Note the distinction between the carbon content of the austenite deposited and that of the layers which deposit. If we were to assume that, in undercooling, the carbon content of the layers which deposit is represented by points on the prolongation of *AE'E* (Fig. 3), it would still remain true that the dissolved carbon content of the austenite in those layers should as a maximum be represented by *SE*, for *SE* represents the maximum solubility of carbon in austenite, the solubility when cementite is bodenkoerper; and if the depositing layers contain more carbon, then that excess should be present as free cementite, or as graphite resulting from its graphitization.)

In short, the difficulty of explaining the persistence ungraphitized of so much cementite would be reduced to an absolute minimum by assuming that the observed arrest temperature, the crest of the recalescence, 1,123°, is the eutectic temperature, and it is reduced nearly to a minimum by assuming that it is only very slightly below the eutectic temperature, say 2° below it.

The argument is rash. For if, as it implies, 0.484 per cent. of carbon in free cementite which existed at the time of the arrest may persist ungraphitized, then the antecedent improbability that a somewhat larger quantity, say 0.584, has so persisted is certainly not great. But if 0.584 has persisted, and if no considerable graphitization occurred after the arrest, then the dissolved carbon of the austenite at the arrest time was 2.184 — 0.584 = 1.6 per cent. But in that case, if, as we assume, the dissolved carbon of the austenite at the eutectic temperature is 1.7 per cent. or 0.10 per cent. more than the dissolved carbon content of the austenite at the arrest, 1,123°, then the eutectic temperature must be 50° above 1,123°, because the slope of *SE* calls for a rise of 5° for each 0.01 per cent. increase in dissolved carbon, or, in short, it must be 1,173°. In short the reasoning is rash, because an only slight and permissible change in one of the premises brings us to a result widely different.

It may be that his ground for setting *EB* at 1,125° is that, granted that *E'B'* is at 1,135°, and further that the horizontal gap between the liquidus parts *B'D'* and *BD* of *GK* and *SK* should form a smooth continuation of the gap between the parts *S'E'* and *SE* of *GK* and *SK* in the solid, then *EB* must needs be some 10° below *E'B'* in order to bring these two gaps thus into match. If so, then 1,125° should be taken not as the observed but as the antecedently probable position of the eutectic temperature based on the position assigned to *S'E'*, which position itself is none too well established.

Though it is true that the sharpness and extent of the rise of temperature, 23°, and the steadiness of the arrest at the peak temperature are more suggestive in this case, No. 4, than in Nos. 1, 2, and 3, of arrest temperatures below the true eutectic, yet the mere fact of this greater undercooling would in itself tend to sharpen and increase the rise of temperature. The suggestions, then, that the true eutectic temperature is between 1,123° and 1,130° are indeed interesting, but are they more than suggestions?

(26) (I.) 1,130° *must be Above EB.*—From the fact that the combined carbon content of alloy of No. 3, which, after cooling to 1,123°, recalesced to 1,130, is less than *E*, viz.: 1.414 per cent. instead of 1.7, Prof. Ruff infers that 1,130° must be above the eutectic temperature, referring the presence of 0.114 per cent. in excess of *E'* (1.414 — 1.30 = 0.114) to the carbon content of the molten reaching *SK* towards the end of solidification.

(26A) Serious gaps appear in the reasoning. For all that I now see, the phenomena are explicable as easily on the hypothesis that 1,130°, the recalescence crest of alloy 3, is below the eutectic temperature *EB*, as on Prof. Ruff's hypothesis that it is above *EB*. Let us run through the course of solidification, first on the assumption that 1,130° is above *EB* and then on the assumption that it is below *EB*. On the former assumption, the course of the molten is somewhat as shown by *P'Q'B*, Fig. 5. Assume undercooling of the austenite (see 18) so that the temperature and carbon content of the molten slide down *P'Q'* till they reach *Q'B'D'* at *Q'*, 1,123°. From this point on the heat due to the precipitation and graphitization of the cementite made metastable by crossing to below *Q'B'D'*, added to the latent heat of the simultaneous precipitation of austenite which has been occurring from *P'* to *Q'*, and still continues, suffices first to retard and then to arrest the fall of temperature, all as shown in Gutowsky's curve No. 3, and then to cause a rise to 1,130°. The precipitation of austenite continues to enrich the molten in cementite faster than the metastability of this excess of cementite leads to its precipitation and graphitization, with the result that the molten increases in dissolved cementite content, and towards the end reaches *B*. Thus the temperature and carbon content of the molten follow a path like *Q′B*, solidification naturally accelerating and completing itself on reaching *B*.

The arrival of the molten at *B* causes it to deposit layers of carbon content *E*, 1.7 per cent.; but yet the fact that the average combined carbon content of the whole, 1.414 per cent., is far less than *E*, is, indeed, only slightly above *E′*, 1.3 per cent., is very easily understood. For (A) the earliest deposited layers, deposited before selective freezing had enriched the molten materially, were, of course, very poor in carbon; and diffusion has had insufficient time to enrich them very greatly; and (B) all the intermediate layers deposited during the progressive enrichment of the molten up to *B* have been deposited with a dissolved carbon content less than *E*, 1.7 per cent., and these, too, have been only incompletely enriched towards *E* by diffusion during the limited time.

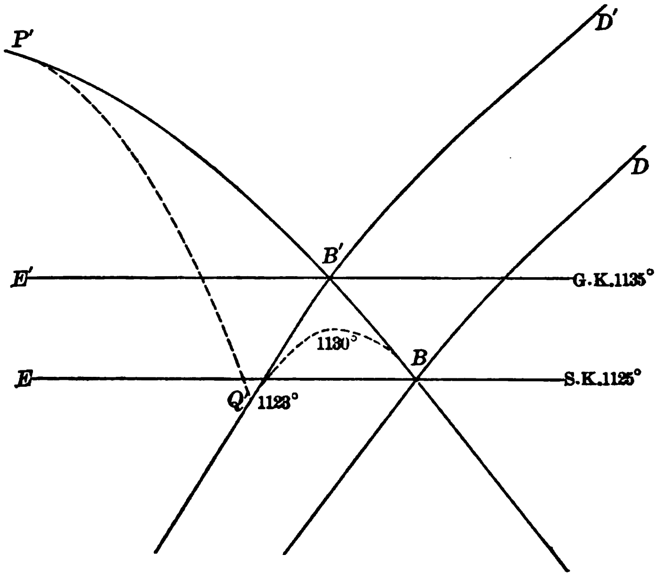

FIG. 5.—THE COURSE OF THE MOLTEN DURING THE SOLIDIFICATION OF GUTOWSKY'S CAST-IRON NO. 3, ON THE ASSUMPTION THAT THE EUTECTIC TEMPERATURE IS 1,125°. (24.)

(27) *The Eutectic Temperature may be Above* 1,130°.—Next assuming that the eutectic temperature is above 1,130°, say at

1,135°, let us represent the course of the molten of alloy No. 3 with a very different temperature scale, as in Fig. 6, assuming the same temperature-course as found by Gutowsky and shown in Fig. 5. With this temperature assigned to *EB*, the observed course of the molten recorded by Gutowsky, Fig. 4, agrees with propositions 12, 17, and 18, as we shall now see. Admitting the undercooling called for by proposition 18, the molten crosses *GK* (*D′B′Q′*) at *Q′*, when the metastabilizing, precipitation, and graphitization of cementite suffice first to retard and then to arrest the cooling at the observed temperature of 1,123°. Because, as before, the precipitation of carbon-poor austenite enriches the molten in carbon faster than this metastabilizing and precipitation of cementite can rob it of carbon, the dissolved carbon content of the molten progressively increases

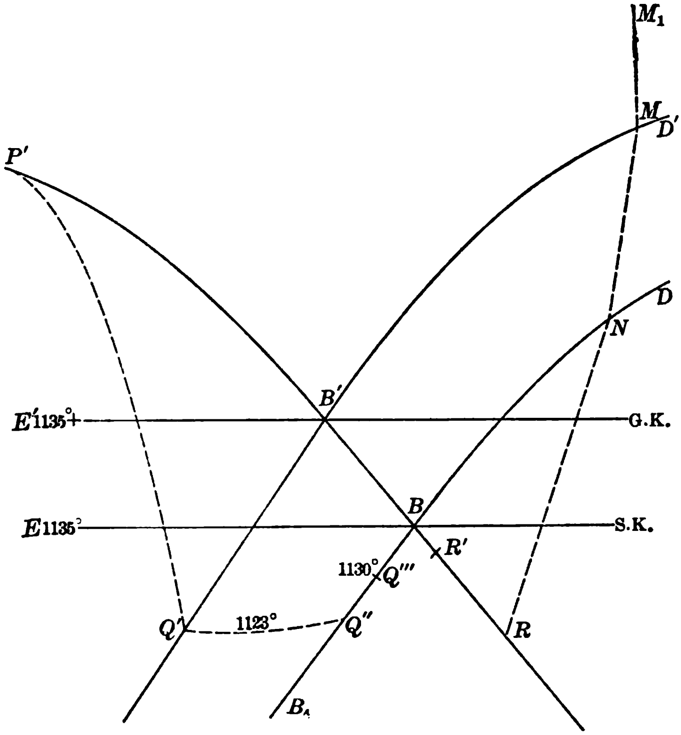

Fig. 6.—The Course of the Molten during the Solidification of Gutowsky's Cast-Iron No. 3, on the assumption that the eutectic temperature is 1,135°. (25.)

from *Q'* towards *Q''*, and with it increases the carbon content of the austenite of the layers precipitating from it, till the molten reaches *SK* (B_4BD) by a path like *Q'Q''*.

The acceleration of the precipitation and graphitization of cementite which now occur causes a recalesence towards *B*, but not fully to *B*, stopping at the observed temperature 1,130°, *Q'''*. The austenite of the layers now depositing has reached *SE*, with a carbon content falling short of 1.7, *E*, by only the very small amount representing the assumed temperature gap between *Q'''* and the eutectic temperature *EB*, but for the same reasons as on the previous hypothesis the average combined carbon content of the austenite as a whole would naturally be only slightly in excess of *E*.'

Indeed, proposition (19) seems to imply that the eutectic temperature must be at least as high as the observed crest of this recalescence, 1,130°, because, if I understand Professor Ruff aright, in the case of a hypo-eutectic alloy such as this present No. 3, the recalescence cannot, in his opinion, rise above the eutectic temperature. Certainly it cannot raise the molten above *APB*, Fig. 2; and apparently with equal certainty it cannot raise it above *SK*, because this would cause a retardation of the precipitation and graphitization of cementite sufficient to check any incipient rise. And if the molten when at 1,130° is neither above *APB* nor above B_4BD (*SK*), it cannot be above *B*, and hence *B* cannot be below 1,130°. Indeed the curve *QB*, which I have been obliged to draw in order to make the observed cooling curve agree with his theory that *EB* is below 1,130°, contradicts his theory set forth in (17) that a material recalescence can hardly occur before the concentration of the molten has reached *SK*. One is tempted to ask whether Prof. Ruff has not overlooked the fact that the recalescence of this alloy No. 3 rises to 1,130°, a fact shown clearly in Gutowsky's Fig. 617, though not referred to in his table. Section 36 gives evidence tending to set the eutectic temperature much higher, even as high as 1,149°.

(28) *A Possible Simplification.*—For the purpose of holding that the width of the gap between *GK* and *SK* in the molten is a smooth continuation of that same width in the solid, it may be important to assume that the true eutectic temperature, *EB*, is 10° below the "near-eutectic temperature," *E'B'*, and that

this is at 1,135°. Yet the phenomena immediately before us seem open to the interpretation that the true eutectic temperature is at least as high as 1,135°, and that the troughs of the recalescences of alloys 1 to 5 represent undercooling below that eutectic temperature. Professor Ruff's extremely ingenious theory may be true, with its "near-eutectic" and true eutectic temperatures, and its view that the undercooling in hyper-eutectic alloys is to between these two temperatures, but in hypo-eutectic alloys to below the lower of them; but if the simpler theory that the governing temperature is the true eutectic temperature, that such undercoolings as occur are undercoolings below it, and such recalescences as occur are attempts to reach it,—if this theory suffices to explain the phenomena at hand, its simplicity would seem to offer a certain advantage, at least in view of the existing conflicts of data concerning the hyper-eutectic liquidus.

Therefore let me run through the phenomena on this simpler theory. On it, and yet adopting Professor Ruff's propositions so far as they do not conflict with simplification, the conditions would be somewhat as follows:

The true eutectic temperature is at least as high as 1,135°. (See 39.)

(29) All that we can infer as to *B′* from the curve of *B′D′* of Ruff and Goecke, and from their experiments tending to show that *BD* is not at the left of *B′D′*, is that *B′* is probably not far above *B*.

In the hypo-eutectic irons solidification begins with the precipitation of primary austenite, say along *P′Q′* Fig. 6, followed on crossing the undercooling prolongation of *B′D′* at *Q′* by the partial precipitation of that part of the cementite thereby made metastable, which graphitizes more or less rapidly according to the temperature. But because the precipitation of carbon-poor austenite enriches the molten in cementite faster than the graphitization of the cementite metastabilized by the passage to the right of *Q′B′* impoverishes it, the carbon content of the molten moves to the right, say along *Q′Q″* till *SK*, BB_4, is reached, say at *Q″*. The starting of the carbon content of the molten to cross *SK* (B_4B) accelerates the precipitation of cementite, the graphitization of which goes on with a rapidity proportional to the existing temperature, and this acceleration of precipitation and graphitization of cementite, added to the continuing solidification of austenite, as both now precipitate eutectically, brings the heat evolution to a rapidity which suffices to cause a recalescence, say to Q''', all as already set forth in section 25.

But while the progressive enrichment of the molten in cementite is progressively raising the dissolved carbon content of the austenite of the layers which that molten deposits from $S'E'$ towards and finally to SE, yet in the layers already solid the opposite is continually occurring. In all those austenite layers deposited after the molten has crossed to the right of GK ($Q'B'D'$) some graphite is present, born of the graphitization of the cementite metastabilized by that crossing.

In any given layer, after the deposition of the next succeeding layers has isolated it from the influence of the molten, the graphite thus present is the bodenkoerper. It is indeed true that any free cementite which has precipitated because metastabilized by the crossing to the right of $Q'B'D'$, and has thus far resisted graphitization, would within the sphere of its influence compete with this graphite in the attempt to act as bodenkoerper. But this is a waning influence, because this cementite is continually graphitizing with a rapidity dependent both on the existing temperature and on the quantity of surface which the cementite and the co-existing graphite offer to the solid austenite in which they are bathed. For the solubility of carbon in contact with graphite particles being less than that in contact with cementite particles, the graphite is continously sucking or precipitating out of this austenite all near-by carbon in excess of $S'E'$, and the cementite is simultaneously dissolving in that same austenite in the Danaidic attempt to keep the carbon content of the environing austenite up to SE. The rapidity of this transfer of the carbon of the cementite through solution in the austenite to the microscopic graphite islets will manifestly depend not alone on the temperature, but also on the extent of surface which graphite and free cementite offer to the austenite bath in which they co-exist, and the surface-extent will vary with the capriciously varying form and distribution of the individual islets of graphite and of cementite. Hence great capriciousness of speed of graphitization is to be expected, and actually exists.

(30) This is the condition of affairs to which I would direct attention, (see (34)), (A) a progressive increase in the dissolved carbon content of the molten, and hence from layer to layer of the depositing solid, with (B) a simultaneous graphitization of the carbon in excess of *S′E′* in the solid isolated from the molten by later deposited layers, and under the over growing influence of the solid graphite. So too with any ungraphitized free cementite of the eutectic. The graphitization is rapid at 1,135°; and though it is retarded greatly and capriciously with fall of temperature, yet it continues at far lower temperatures. To this point I will return. (See 37.)

(31) Turning now to the Gutowsky cases in the concrete, Table I. and Fig. 4, note first the progressive lowering of both trough and crest of the recalescence from 1,130° and 1,138° in No. 1 to 1,093° in No. 5, with progressive fall in the total carbon content. This descent of the recalescence with decreasing carbon content is parallel to the familiar descent in the case of Ar1, and may be referred to like causes, especially to the greater undercooling effect of the larger mass of pro-eutectic austenite, wholly irresponsive to the passage past *EB*.[15] Next note the progressive decrease in the proportion of the total carbon which graphitizes as the carbon content decreases. Thus 79 per cent. of the total carbon graphitizes in No. 1, and only 21 per cent. remains combined, whereas in No. 5 only 13 per cent. graphitizes, and in No. 6 none does. This seems readily explicable by two very simple influences. 1st, the progressive lowering of the temperature of solidification and the great retardation of the graphitization consequent on this lowering; and 2d, the progressive enfeeblement of the graphitization at all lower temperatures through this restriction of the quantity of graphite formed at temperatures at which graphitization is rapid. Graphitization is a process which should accelerate itself. The graphite is a sponge which sucks the dissolved carbon out of the solid austenite, and thus leads the cementite present to dissolve, so that the more graphite is present the faster should be the graphitization of the residual cementite.[16] Witness Gutowsky's finding that, though in the

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[15] See also Howe, *Proceedings of the American Society for Testing Materials*, vol. xi., p. 303 (1911).

[16] *Metallurgie*, vol. vi., p. 739 (1909).

initial absence of graphite the graphitization of cementite was nil during 3 hr., at 1,080° to 1,090°, quite as in Tiemann's[17] experiments, yet a mere prolongation of exposure to this same temperature caused complete graphitization of the whole of the free cementite in 24 hours.

(32) The course of solidification of a hypo-eutectic alloy has just been shown in (29) to agree with this general conception. Turning now to the course of the molten of a hyper-eutectic alloy, it may be conceived as along M_1MNR in Fig. 6. That is to say the molten of carbon content M and originally starting at temperature MM_1 cools without solidification till, on crossing $B'D'$ at M, the cementite thereby metastabilized begins precipitating, and, because of the very high temperature, as it precipitates graphitizes very rapidly. This outfalling of cementite moves the carbon content of the molten proportionally to the left. The undercooling below BD, which Prof. Ruff confines to cases of very rapid cooling, I assume to occur even with Gutowsky's rate of cooling, an assumption which seems to me perfectly admissible. The crossing of BD at N accelerates the precipitation and graphitization of cementite, and the carbon content of the molten shifts more rapidly to the left, striking $P'B'B$ at its prolongation R. The arrival at this line causes the co-precipitation eutecticwise of austenite with the cementite still precipitating, and this addition to the heat evolution suffices to cause a recalescence along RB, say to R'. Thus may be explained the recalescence which occurs in both of Gutowsky's hyper-eutectic alloys.

(33) The variations in the combined carbon content of Gutowsky's alloys explain themselves readily as representing (A) the fact, brought out in column 7 of Table I., that, as the total carbon content decreases, the proportion of that total carbon which graphitizes decreases; and this in turn readily explains itself through (B) the fact that the smaller the total carbon content, the greater is the undercooling, and consequently (C) the lower the temperature at which the cementite, initially dissolved in complete stability, precipitates and thus becomes graphitizable; (D) that the lower this temperature the slower

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[17] *The Metallographist*, vol. iv., p. 319 (1901). Tiemann's iron contained 0.255 per cent. of graphite initially, which did not increase appreciably during the heatings.

is graphitization; and (E) that the more graphite is present, the faster is graphitization. It is necessary to bear in mind both that cementite can graphitize after solidification has completed itself, indeed at much lower temperatures; and that this graphitization is capricious, both of which points I touch on elsewhere. (See 36 and 37.)

(34) Thus there is a race between two opposing tendencies, (A) the progressive enrichment of the molten, and the consequent enrichment of the austenite layers depositing, tending to increase the final resultant combined carbon content, and (B) the tendency to lower that combined carbon content through graphitization. The enrichment of the molten we have seen in §§ 26 and 27 tends to raise the dissolved carbon content of the successively deposited layers up to *SE*, and on reaching *SE* to lead to the birth of eutectic throughout the remainder of solidification, an influence tending to raise the combined carbon of the whole. On the other hand graphitization tends to lower that combined carbon content to *S'E'* for the existing temperature, 1st by graphitizing all free cementite and thus eliminating it as a bodenkoerper, and 2d by precipitating and graphitizing all dissolved cementite in excess of *S'E'* for the existing temperature.

Thus the various Gutowsky data explain themselves readily on this simplified hypothesis that the true eutectic temperature is as high as 1,135°, and that all the undercoolings are undercoolings below the true eutectic temperature, if we admit that with such a rate of cooling as Gutowsky's undercooling may pass below *BD*. The question then is, "Is this admission justifiable?"

(35) As regards the effect of a rapid cooling, note that on one hand it tends to lessen the quantity of combined carbon by restricting the diffusion of the carbon back from the later deposited and richer austenite layers and from the eutectic austenite and cementite, if any, to the earlier deposited and poorer layers, so that at the end the austenite on an average of its whole will contain less dissolved and hence perfectly stable ungraphitizable cementite, and more of the cementite will be present, either metastable at the right of *Q'B'D'*, and thus precipitable and graphitizable, or even in the molten on B_4BD, and hence quickly precipitating and thereby graphitizable. But on the

other hand rapidity of cooling lessens the opportunity for graphitization of the cementite made free, both by shortening the time available and by leading to a lower birth temperature. Thus by increasing the undercooling of the austenite it lowers the temperature at which $Q'B'D$ and B_4BD respectively are crossed, *e. g.*, from B' and B to Q' and Q'', *i. e.*, the temperature at which the precipitation of cementite begins and that at which it accelerates critically.

Experience proves abundantly that, under all usual conditions, this graphitization-impeding effect of rapidity of cooling outweighs its contrary effect in restricting diffusion.

(36) *Capriciousness of Graphitization, i. e., Important Influence of Unnoticed Variations in the Conditions.*—From the fact that the graphitization of cementite should occur, at least in large part, through the action of the graphite nuclei in sucking the carbon out of the austenite, and thereby leading to the dissolving of the free cementite, that this sucking action occurs at the surface of the particles of graphite, and that the dissolving of the cementite occurs at its own surface, and, further, that the extent of surface of each is likely to vary capriciously with the exact conditions of crystallization, we naturally expect the rate of graphitization to be capricious. Our general knowledge of the subject agrees with this expectation. But specific instances are at hand. Thus, Ruer and Iljin,[18] experimenting on two apparently like pieces of gray cast iron, initially with 0.50 per cent. of combined carbon, Nos. 1 and 5 of Table II., heated both to 1,100°, held them there for 3 hr., and then cooled them during 3 hr. to 800°. Thus far all the conditions seem to have been identical. At 800° the dissolved carbon content should, according to Ruer and Iljin's line, have fallen to about 0.75 per cent., and the rest of the combined carbon naturally should have existed as free cementite. Arriving thus at 800°, No. 5 was cooled pretty rapidly, in from 10 to 15 min., to room temperature; No. 1 was held at 800° for 12 hr. and then quenched. The one held 12 hr. at 800° had 1.27 per cent. of combined carbon, practically that which it should have held in solution at 1,100°, so that there had been no graphitization either during the 3 hr. cooling to 800° or during the 12-hr. sojourn at

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[18] *Metallurgie*, vol. viii., p. 100 (1911).

TABLE II.—*Capriciousness of Graphitization in the Experiments of Ruer and Iljin.*

No. 1.	No. 5.	No. 7.	No. 8.
1,100° for 3 hours. ↓ 3 hours. 800°		1,100° for 3 hours. Quenched. Room T°	
800° for 12 hours. Quenched. Room T°	800° ↓ 10 to 15 min. Room T°	800 for 4 hours.	900° ↓ 3 hours. 800 for ½ hour.
		↓ 2 hours. 700° for 1 hour. Quenched. Room T°.	
Combined Carbon. Per Cent. 1.27	Combined Carbon. Per Cent. 0.73	Combined Carbon. Per Cent. 0.93	Combined Carbon. Per Cent. 0.57

800°. The other, cooled somewhat quickly immediately on reaching 800°, contained only 0.75 per cent. of carbon, or that corresponding to the solubility of carbon at 800°, on Ruer and Iljin's line. Thus in the former case during 3 hr. cooling plus 12 hr. sojourn at 800° no graphitization occurred, whereas in the latter case complete graphitization of the carbon in excess of *S'E'* occurred during the 3 hr. cooling, without any sojourn at all; or, in short, the graphitization was most capricious.

It is true that they try to explain away the discrepancy between their Nos. 1 and 5 by imagining that the 0.54 per cent. extra graphitization of the combined carbon of No. 5 occurred during the 10 to 15 min. slow cooling from 800° to the room-temperature, explaining that while graphitization does not take place between 1,100° and 800°, it does below 800° if there has been a sojourn above 800°. But this explanation should be accepted very cautiously, (A) because the graphitization of as much as 0.54 per cent. of combined carbon in the almost complete absence of silicon in a 15-min. cooling from 800° to the room-temperature is far more rapid than our general knowledge of the conditions of graphitization leads us to expect; (B) because this explanation fails completely to explain the very great differences between their Nos. 7 and 8, which apparently had equivalent sojourns above 800° and identical conditions below 800°; and (C) because in my pure cast-iron described in this section, though there was long exposure to temperatures between 800° and 1,000°, graphitization on subsequent very protracted exposure to temperatures below 800° was extremely slow.

Again in their cases 7 and 8, the conditions were almost identical, as shown by the parallel columns in Table II., yet the ungraphitized carbon was 63 per cent. greater in No. 7 than in No. 8.

Again a specimen of very pure cast-iron, prepared under my directions by my assistant, A. G. Levy, by melting Professor Burgess's pure electrolytic iron under an excess of sugar charcoal, certainly seems to have retained its combined carbon much more tenaciously than the irons of Ruer and Iljin, having no less than 1.14 per cent. of combined carbon out of a total carbon content of 3.494, after prolonged exposure to temperatures both above and below 800°. The treatment of this specimen for the purpose of graphitizing the cementite consisted first of a 2-hr. cooling from 1,100° to 900°–1,000°, followed by a moderately slow cooling to the room-temperature, then of five slow coolings from 815° and from 790°, aggregating at least 11 hr., in order to graphitize the pro-eutectoid cementite precipitating at these temperatures, and finally in very long holdings, 79.5 hr. between 635° and 710°, and 118.5 hr. between 700° and 750°, for the purpose of graphitizing the cementite formed on cooling below the "near-eutectoid" temperature, *i. e.*, the junction of *S'E'* and *GOS.* Though at the end of the 79.5 hr. holdings the combined carbon was reduced to 1.14 per cent., yet some prominent crystals of cementite persisted ungraphitized even till the end of the additional 118.5 hr., as shown in Fig. 8. They were shown to be cementite by darkening with sodium picrate. The persistence of the initial ingot structure is shown by the well-marked rectangular areas in Fig. 9 of this same specimen after the six slow coolings. The white dendrites are evidently the relics of the layers first to solidify, and therefore poorer in carbon than those which solidify later. The persistence of this dendritic structure shows how slow diffusion may be. Other instances of very tardy graphitization are given in Appendix I.

In view of this great capriciousness of the rate of graphitization, the quantitative determinations of this rate by Ruff, based on these few cases of Gutowsky's, should not, I believe, receive great weight, especially because the variations in the silicon of the Gutowsky irons were such as to contribute to the decrease of graphitization with decreasing carbon content. Thus his hyper-eutectic irons had 0.09 per cent. of silicon, whereas his hypo-eutectic ones had only from 0.045 to 0.064. Again though the slower cooling of Carpenter and Keeling's alloy No. 20 of Table I. should have favored graphitization, it had no graphite, though the closely corresponding No. 5 of Gutowsky had 0.282 per cent.

Again the combined carbon content in Carpenter and Keeling's irons cooled under apparently like conditions varied capriciously, as shown by Nos. 14 to 20 in Table I. Though there is indeed a general decrease in graphitization as the carbon content falls, yet the irregularities in this decrease are greater than we should expect from unrecorded variations in the conditions.

Like variations occur among the Wüst cases in Table I., but these seem referable to the capricious variations in the temperatures at which the arrests occur, which variations themselves are unexplained as noted below.

Further indications of considerable capriciousness are found (1) in the much higher arrest or recalescence temperatures of some of Carpenter and Keeling's alloys, *e.g.*, Nos. 15, 18, and 19, than in the corresponding ones of Gutowsky; in the irregularity with which this arrest temperature varies with decreasing carbon content in the Carpenter and Keeling alloys; and in the imperfect accord between Gutowsky's first and second series. The more rapid cooling in his second series would be expected to favor undercooling; yet the undercooling seems on the whole less than in his first series, for like carbon content.

The Wüst cases in Table I. give like evidence. First their arrest temperature even in hypo-eutectic irons rises to 1,149°, or far above any corresponding temperature reached under like conditions in the Gutowsky data and reminding us that Wittorf puts the "near-eutectic" temperature as high as 1,180°. Second they vary capriciously among themselves, the two lowest in carbon, Nos. 27 and 28 with 3.02 and 2.94 per cent. of carbon respectively, have their arrests at 1,135° and 1,132°, or well above the arrests of alloys much richer in carbon, e.g., Nos. 22 and 26, with 4.04 and 3.29 per cent. of carbon respectively, and with arrests at 1,122° and 1,112°.

The alloy richest but one in carbon, No. 22, has its arrest at a temperature much below that of any of the others except No. 26. The only hyper-eutectic alloy, No. 21, has its arrest temperature lower than the alloy poorest in carbon, No. 28, and lower than four other hypo-eutectic alloys, Nos. 23, 24, 25, and 27. In some of these cases the variations in the rate of cooling go towards explaining the discrepancies, but not the discrepancy between Nos. 22 and 23. Note that though Nos. 24 and 25 have almost identical carbon content, and though the much slower cooling of 25 should tend to raise its arrest temperature, in fact the temperature is 3° lower than in No. 24.

The interpretation of this evidence should be extremely cautious. The most striking thing about it is that, whereas each of the Gutowsky series is concordant taken in and by itself, with the arrest or recalescence temperature as the case may be falling progressively with the carbon content; and whereas the anomalies in the Heyn and Bauer cases may readily be due to the variations in silicon content, yet the Carpenter and Keeling and the Wüst results are not concordant *inter se*, the arrest temperature having little relation to the carbon content. Even if we were to challenge the Carpenter and Keeling results because the cooling curves themselves are not given, the Wüst difficulty would remain, for these cases rest on legible inverse-rate curves, with very sharp retardations.

In view of the antecedent probability that the eutectic temperature should fall with decreasing carbon content; of the continuous fall in each of the Gutowsky series; of the apparent very great care with which he worked; and of the possibility that these earlier investigators may have overlooked some needed precaution; one comes away with the impression that the law is probably true, but that the unnoticed variations in the conditions may easily obscure it under the intrinsically difficult observational conditions. We can at least say this: that in each series in which there is a carbon content less than 2.74 per cent., this further lowering of the carbon content is accompanied by a marked lowering of the arrest temperature. Further evidence is awaited.

(37) *Graphitization Continues at Temperatures far below the Freezing-Range.*—If we accept the line of Ruer and Iljin, then it is clear that, in Gutowsky's alloy No. 1, graphitization continued at least down to 1,000°, for in no other way can we explain how its combined carbon content comes to be so low, 1.03 per cent. For if as long as the cementite is dissolved in the austenite in quantity not to exceed *S'E'* it is perfectly stable and ungraphitizable, then graphitization could not reduce the combined carbon content to 1.03 per cent. as it did, till the temperature had sunk to 1,000°, at which the solution and hence ungraphitizable carbon content for the first time falls as low as 1.03, only that in excess of 1.03 being graphitizable.

Again in the malleablizing process the combined carbon content often becomes negligible. In the case given in Table III. it is only 0.022 per cent. Benedicks[19] too graphitized completely the pearlitic cementite of a gray-iron by seesawing at temperatures which rose at the highest to but little above 800°. Further I have collected three trustworthy cases in which commercial cast-iron contained no combined carbon, together with one in which it had only 0.04 per cent.[20]

But the ungraphitizable dissolved carbon content evidently cannot fall so low at any point on *S'E'* above *S'*, the *GK* or "near-eutectoid" temperature, *A'1*, at which *S'E'* cuts *GOS* at carbon content about 0.70 per cent. On cooling past *A'1* a transformation parallel to that which occurs in cooling past *A1* should occur; *i.e.*, the austenite under the domination of graphite as bodenkoerper should here break up into a conglomerate of ferrite and graphite, just as, when cementite is bodenkoerper, austenite breaks up at *Ar1* into ferrite and pearlitic cementite. Hence when we find the combined carbon less than the *A'1* carbon content, say 0.70 per cent., we infer that graphitization has continued to below *A'1*.

Ruer and Iljin's line implies that *A'1* is at least as low as 775° even if we accept the high position assigned to *GOS* by Goerens and Meyer.[21] This then implies that, at least in the presence of silicon as in the manufacture of malleable cast-iron, graphitization continues habitually at least down to 775°.

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[19] *Metallurgie*, vol. v., p. 42 (1908).

[20] *Trans.*, xxxix., 58 (1908).

[21] Goerens and Meyer, *Metallurgie*, vol. vii., pp. 307–312 (1910).

Ruer and Iljin's paper has confirmatory internal evidence (their Table I.) tending to show that *A'1*, the near-eutectoid temperature, is between 700° and 800°; for on reheating graphitic cast-irons in which the combined carbon content had been brought very low, there was marked increase in the combined carbon when the reheating temperature was 800° but none when it was 700°. At *A'1* re-combination up to *S'E'*, 0.7 per cent. of carbon should occur, the graphite present dissolving in the austenite to that extent, but we hardly look for such recombination below *A'1.* It may indeed be true that external graphite will dissolve in iron between *A1* and *A'1*, the small quantity of pearlitic cementite present dissolving to form a small quantity of austenite on rising past *Ac1*, and the external graphite cementing that austenite. But internal graphite present so abundantly as in these irons of Ruer and Iljin would be expected to act so efficiently as bodenkoerper that, at all temperatures below *A'1*, the solubility of carbon in the presence of graphite being nil here, it would precipitate all dissolved carbon as fast as it dissolved. Further evidence on this point is needed.

But even in Ruer and Iljin's alloys nearly free from silicon (0.05 per cent. of silicon with 0.21 per cent. of manganese) the combined carbon content fell to 0.27 per cent., which by the reasoning just set forth, shows that graphitization must have continued at least down to *A'1*, *i. e.* at least down to 775°.

And whatever reasonable position we assign to *GOS* and *S'E'*, the frequent low combined carbon content of very graphitic malleable castings, the occasional like combination in gray cast-iron, the low combined carbon of Ruer and Iljin, and the complete graphitization by Benedicks, show with mathematical certainty that graphitization continues down to temperatures very far below the freezing range unless our present theories are far wrong.

That graphitization goes on with very considerable rapidity in the presence of silicon at temperatures below the freezing range is shown by evidence from several courses.

Heyn and Bauer[22] and Goerens and Gutowsky[23] show this by determining the degree of graphitization which occurred at various stages of cooling above, in, and below the solidification range. This they did by allowing a series of specimens to cool slowly to a series of known temperatures in and near that range, and then interrupting the cooling by quenching in water, so as to arrest the graphitization at the stage which it had reached at each temperature. Chemical analysis of the quenched specimens showed in every series that graphitization had gone on to an important degree below the solidification range.

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[22] *Stahl und Eisen*, vol. xxvii., pp. 1565-1571 and 1621-1625 (1907).

[23] *Metallurgie*, vol. v., p. 145 (1908).

This is further indicated by the reported industrial observation that "castings made in chill moulds show—no signs of chilled structure, provided the castings are removed immediately after casting."[24] Had they been left in contact with the chill they would have remained chilled. Because the conditions up to the moment of removal are identical whether the casting is left in contact with the chill or is removed, and because if left in contact with the chill its face is chilled, *i. e.* graphiteless, it follows that at the moment of removal the face of removed castings is graphiteless. But because it is solid when removed from the mold, and because after the ensuing relatively rapid cooling it is found to be gray, it follows that graphitization in these cases is habitually rapid below the freezing-range.

T. D. West[25] found that to reheat chilled cast-iron plates to temperatures well below the melting-point by pouring molten cast-iron against them caused slight graphitization, as inferred from the appearance of the fracture. With further reheating the graphitization increased, till when the temperature of the face reached the melting-point, the face of the plate had been brought about one-quarter way towards the condition of gray iron. In each case the reheating was rapid and brief.

He further finds that if a cast-iron casting, after the outer part has solidified and cooled at a normal rate to somewhat below the eutectic freezing-point and thus become graphitic, is suddenly withdrawn from the mold and quenched in water, the inside is of white iron. The fact that the inside of the casting is white though the normal slow rate of cooling has extended below the eutectic freezing point, shows that it solidified as white iron. The graphitization of the outside must have occurred in that phase of the cooling by which the gray outside had outrun the white interior at the moment when the quenching occurred. Because the amount by which the cooling of the outside has outrun that of the inside must be very slight, we infer that this graphitization occurerd in a very narrow range of cooling and presumably below the eutectic freezing point. His further ingenious experiments show that, if the slow cooling had not been interrupted thus, the interior would have been very gray.

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[24] P. Munnoch, *Iron Age*, vol. lxxxix., p. 1414 (June 6, 1912).

[25] *The Foundry*, vol. xl., pp. 245-251 (June, 1912).

(38) *How Dominant is the Influence of the Bodenkoerper ?*—There is one fact which is not easily reconciled with one of the foregoing conceptions and two of the foregoing assumptions. The conception is (1) that the nature of the bodenkoerper, *e.g.*, whether it is free cementite or free graphite, determines whether the dissolved carbon content of austenite shall be represented by *SE* or *S′E′*. The assumptions are (2) that *SE* is truly represented by the determinations of Gutowsky, and (3) that *S′E′* is truly represented by those of Ruer and Iljin.

The fact difficult to reconcile with the foregoing is that the difference between the conditions in certain of Gutowsky's experiments and the conditions in Ruer and Iljin's seems insufficient to explain why cementite should have dominated completely in the former, and graphite completely in the latter, bringing the combined carbon only to *S′E′*.

In Gutowsky's No. 42 with gray iron and 43 with white iron, in spite of the final presence of over 2 per cent. of internal graphite the dissolved carbon content of the austenite followed the line *SE* with great precision; whereas in Ruer and Iljin's in the same range of temperature and with about the same quantity of internal graphite it followed the line *S′E′*. The sole material difference in conditions appears to be that Gutowsky's cylinders of cast-iron were imbedded in a carburizing substance, while Ruer and Iljin's were not. They were enclosed in sealed quartz tubes. There were present in these tubes, 1st the negligible quantity of air present at the time of sealing, and 2d whatever gases were evolved by the iron itself during the prolonged heatings.

Taking these results at their face they seem to teach that the influence of external animal charcoal in Gutowsky's Nos. 42 and 43 has completely dominated and indeed effaced the influence of about 2.1 per cent. of internal graphite. But this is so improbable antecedently that we hesitate to accept it. For this internal graphite, at least in No. 43, must have been very finely divided. Therefore, distributed as it was throughout the whole mass of the austentite and thus offering enormous and most intimate surface exposure, its efficiency as bodenkoerper certainly ought to be much greater than that of the external animal charcoal simultaneously present, which could at best offer only slight and not intimate contact.

Two escapes from this situation offer themselves. (1) The assumption that the influence of internal graphite, even when finely divided and distributed internally through the iron, in dragging the dissolved carbon content back from *SE* towards *S′E′*, is so feeble that it is completely negligible when it competes with the influence of external amorphous carbon in cementing the austenite up to *SE*. This path of escape from the difficulty is certainly a most thorny one.

(2) The influence of the bodenkoerper may be altogether secondary to that of some unnoticed difference between the conditions of Gutowsky and those of Ruer and Iljin, a difference which led the former always to *SE* and the latter always near to *S′E′*. To throw light on this I purpose repeating both sets of experiments under identical conditions.

In any event, till this matter is cleared up the results of Ruer and Iljin should be received with some caution for two reasons. (A) for each temperature they differ between themselves by from 0.10 to 0.16 per cent., or from about ⅓ to about ½ of the gap between their *S′E′* and Gutowsky's *SE*. (B) Moreover there is reason to fear that the results of the "absorption" method which they followed may be either too high or too low. This method consists in holding graphitic iron at a temperature for which the *S′E′* carbon content may be assumed safely to be greater than the initially dissolved carbon content, in order to allow it to absorb the internal graphite till it reaches *S′E′* and thereby saturates itself. The results thus reached may either be (1) at the left of the true *S′E′* because of insufficient time to complete the absorption to that line; or (2) they may be at the right of the true *S′E′*, because on unnoticed petty falls of temperature, free cementite would precipitate, which on the subsequent unnoticed rise would compete with the internal graphite in carburizing the austenite, so that their joint action would carburize it to a point between true *S′E′* and true *SE*.

Moreover any free cementite not dissolved in carburizing the austenite would be counted in with the combined carbon and be found by Ruer and Iljin. Against (2) it may be urged that though a like precipitation of free cementite might bring the *SE* determinations of Gutowsky to the right of the true *SE*, yet if we may judge from the great concordance of his results it did not. And if it did not in his case, why should it in Ruer and Iljin's?

The "precipitation" method is in a better position, because, while its results may be too high, they can hardly be too low; hence the lowest trustworthy results reached by this method should be taken as the nearest experimental approach to the true $S'E'$. This method consists in holding graphitic iron at a temperature at which its initial dissolved carbon content is more than enough to saturate it, *i. e.*, exceeds $S'E'$, in order to allow the excess of dissolved carbon to precipitate out and to graphitize, then cooling rapidly, and determining the combined carbon. The results of this method may indeed be too high (A) because the precipitation of the excess of the dissolved carbon may lag, and (B) because the graphitization of the cementite thus precipitated may lag; but it is not easy to see how they can be too low.

Unfortunately in the absence of silicon this graphitization of cementite is extremely sluggish, as is shown by the results reached with it by Ruer and Iljin and by myself and Levy ((36) (and Appendix I). In view of this the most promising way seems to be to determine $S'E'$ by the precipitation method with two different percentages of silicon, in order to confirm or disprove the current belief that silicon does not shift the position of $S'E'$ but only hastens the arrival at it. If this belief is confirmed, then the position of $S'E'$ thus found would be its true position. If it is disproved, there is still the possibility of finding the true position of $S'E'$ in silicon-less irons by extrapolation.

(39) *Conclusion.*—We have now seen in (36) that the unexplained variations in the cooling curves are very great, that the two series of observations on this point by Gutowsky do not agree; and that the rate of graphitization is very capricious. In (37) we have seen that graphitization continues at temperatures far below those to which Prof. Ruff seems to con-

fine it. In (38) we have seen difficulty in reconciling the Gutowsky and the Ruer and Iljin lines with the assumed dominance of the bodenkoerper, all of which seem essential premises underlying Prof. Ruff's reasoning. In view of these facts the data as to arrest temperature and graphitization in the single Gutowsky series used by Prof. Ruff seem to me wholly insufficient for a firm foundation on which to base calculations as to the equilibrium position of the eutectic temperature.

The new data offered in this paper have been reached in experiments carried out in part under a grant from the Carnegie Institution of Washington, in the Metallurgical Laboratories of Columbia University. For the analyses we have to thank Messrs. Booth, Garrett & Blair, and for the electrolytic iron Prof. C. F. Burgess.

APPENDIX I.—A CASE OF SLOW GRAPHITIZATION ABOVE 800°.

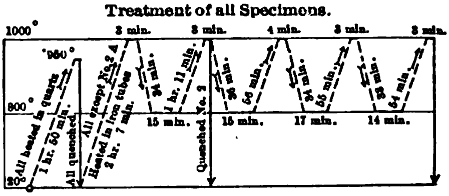
The experiments here described made by A. G. Levy under my direction consisted in exposing to certain oscillations of temperature a specimen of malleable cast-iron, No. 1, and four cylinders, Nos. 2, 3, 4, and 5 of gray cast-iron made by cementing and melting Prof. C. F. Burgess's electrolytic iron. These four cylinders were cut out from a single small ingot, called "Original" in Table III. In making this ingot the electrolytic iron was melted with an excess of pure sugar charcoal in a graphite crucible brasqued with sugar charcoal, bound with a little liquid asphalt and vaseline. The crucible was highly heated in a gas-furnace, opened once to make sure that the mass was molten and immediately closed. After further holding it was then cooled to about 1,100° by lessening the gas supply, and thence to between 800° and 900° in 4 hr. by further successive reductions of the gas supply. The resultant ingot is called the "Original" in Table III.

Four cylinders, Nos. 2 to 5, approximately 2 in. long by 0.25 in. in diameter, cut from this ingot were enclosed each in a sealed quartz tube, and reheated to about 950°, when on account of an accident they were all quenched in water. No. 2A (Figs. 10 and 11) represents the condition after this quenching.

Each of these four cylinders was then enclosed in a steel gas-tube closed at each end with a cap. No. 4 was packed with sugar charcoal in its tube, and No. 5 with kish from an iron blast-furnace. The tubes containing Nos. 2 and 3 had no carburizing material, and each was separated from its specimen by means of sheet asbestos.

The specimen of malleable cast-iron, No. 1, was enclosed in a like steel tube.

TABLE III., AND FIG. 7.—*Graphitization and Re-Combination in Certain Cast-Irons.*



	Cast-Iron. Original of Nos. 2 to 5.	Malleable Cast-Iron, Original of No 1.		Cast-Iron No. 2 A. Alone.		Cast-Iron No. 2. Alone.		Cast-Irons.			Malleable Iron No. 1. Alone.
								No. 3. Alone.	No. 4. Sugar Char-coal.	No. 5. Kish.	
Comb. carbon	1.598	0.022	......	....	....	1.554	......	1.130	1.542	1.628	1.030
Graphite	1.356	1.952	......	....	....	0.930	......	0.454	1.040	0.948	0.738
Total carbon	2.954	1.974	......	....	....	2.484	......	1.584	2.582	2.576	1.768
Silicon	0.040	0.717									
Manganese	None.										
Phosphorus	Not det.										
Sulphur	0.035										

NOTE.—For the chemical analyses we are indebted to Messrs. Booth, Garrett & Blair.

These five specimens, each in its tube, were then heated together to 1,000°, cooled slowly to 800°, and reheated to 1,000°, when No. 2 was drawn and quenched with its containing tube in cold water. The remaining ones were then reheated to 1,000°, cooled slowly to 800°, and thus seesawed up and down as shown in Fig. 7, till on reaching 1,000° for the fifth time they were drawn and quenched in water each in its enclosing tube. The cooling in this quenching was not very rapid, as is shown by the fact that Nos. 2, 3, 4, and 5 could be sawed with a hack-saw, though No. 1, the specimen of malleable cast-iron, could not. No. 3 which had become much decarburized also had decidedly less combined carbon than the original. The procedure and results are shown in Fig. 7 and Table 3.

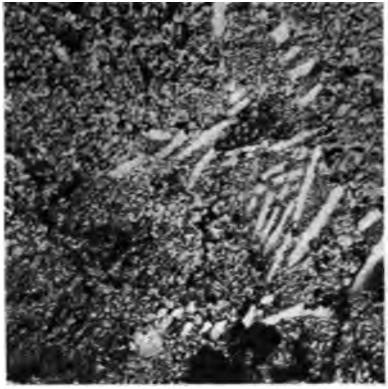


FIG. 8.—PERSISTENCE OF CEMENTITE CRYSTALS IN PURE CAST-IRON AFTER PROLONGED HEATING, INCLUDING 118 HOURS BETWEEN 635° AND 750°. MAGNIFIED 85 DIAMETERS.

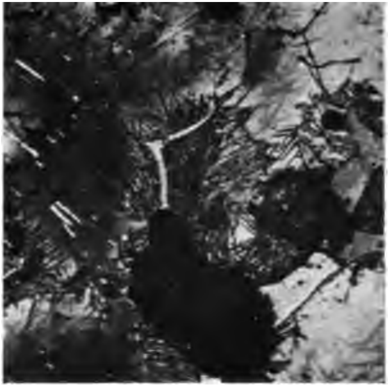


FIG. 10.—PURE CAST-IRON NO. 2A QUENCHED FROM 950°. CONTACT OF EUTECTIC WITH GRAPHITE. FINE NEEDLES ARE PRO-EUTECTOID CEMENTITE. MAGNIFIED 85 DIAMETERS.

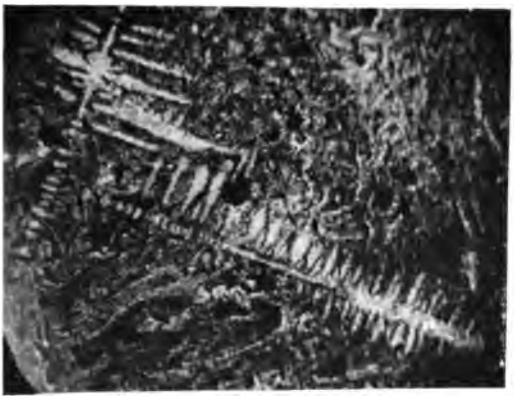


FIG. 9.—PERSISTENCE OF THE ORIGINAL SOLIDIFICATION DENDRITES AFTER SIX SLOW COOLINGS. PURE CAST-IRON ETCHED WITH SODIUM PICRATE. MAGNIFIED 18 DIAMETERS.

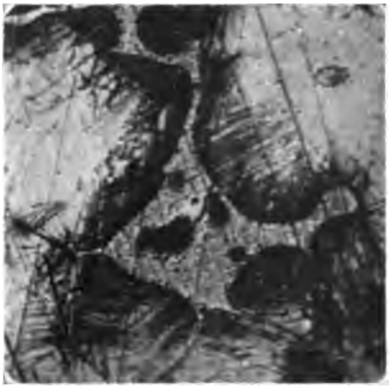


FIG. 11.—PURE CAST-IRON NO. 2A QUENCHED FROM 950°. CONTACT OF EUTECTIC WITH GRAPHITE. FINE NEEDLES ARE PRO-EUTECTOID CEMENTITE. MAGNIFIED 85 DIAMETERS.

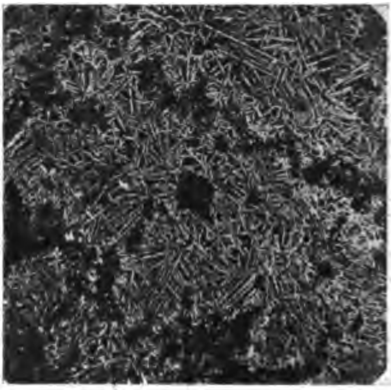


FIG. 12.—PURE CAST-IRON QUENCHED AFTER REPEATED OSCILLATIONS OF TEMPERATURE, FIG. 7. MAGNIFIED 85 DIAMETERS.

Each of the quenched specimens had a very considerable quantity of well oriented cementite needles, part of which may well represent precipitation during the quenching, though part represents the combined carbon present at 1,000° in excess of *SE*, or more likely in excess of some line between *SE* and *S'E'*. The microstructure of Nos. 2, 3, 4, and 5 was almost identical, and is shown in Fig. 12; that of No. 2A, Figs. 10 and 11, had very much fewer fine cementite needles, and instead some rather massive eutectic islands. The massive white islands are in most cases clearly masses of eutectic. Even the arrowhead mass in Fig. 10 is very probably of eutectic. The cementite of this eutectic (Fig. 10) has escaped graphitization both in the original solidification and in reheating. The relative freedom of Nos. 2 to 5 from eutectic suggests that in the successive heatings the cementite of this eutectic has been washed away to an extent hardly to be foreseen, to re-precipitate as fine needles. The immediate contact of graphite with this eutectic in Figs. 10 and 11 is to be noted. Here the acicular regions about the graphite represent the austenite formed at a high temperature along with that graphite from the graphitization of cementite. The needles are the pro-eutectoid cementite which has precipitated out of that austenite in cooling down, as is seen on comparing this structure with Fig. 12.

At the first glance one might take some of these large eutectic masses for cementite, and wonder at the immediate contact of this cementite with graphite, a thing which, after diligent search and inquiry, I have never found save in some micrographs by Prof. Wüst[26] and Prof. Goerens.[27]

The combined carbon content of Nos. 2, 4, and 5 varied only within the limits of 1.542 and 1.628, corresponding roughly to the carbon dissolved between 1,050° and 1,100° according to the line *SE*. Though the total period of exposure to temperatures between 800° and 1,000° was not very great, one certainly would have strongly expected some graphitization of the carbon, reducing the combined carbon towards *SE* for that range, or even below *SE* in view of the large quantity of graphite present. Yet the iron which stayed longest in this range, No. 5, actually had a little more combined carbon than the original. Thus even in the presence of as much as 1.598 per cent. of graphite, free cementite may persist apparently without undergoing even partial graphitization during a very considerable sojourn rising repeatedly to 1,000°.

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[26] *Metallurgie*, vol. iii., Figs. 14, 15, 16, opposite p. 12 (1906).

[27] Introduction to *Metallography*, 1908, Figs. 142, 143, and 144, pp. 197–198.

The malleable cast-iron is in great contrast to these others. Its combined carbon content rose to 1.03 per cent., which agrees fairly with the maxima of Ruer and Iljin's results. Their highest two combined carbon contents on heating gray cast-iron to 1,000° without carburizing material, and then quenching, were 1.02 and 1.07 per cent.

The extreme slowness of the graphitization of my pure cast-iron is to be referred to its low silicon content.

APPENDIX II.—SOLVENT, SOLUTE, AND BODENKOERPER.

When we saturate a common aqueous solution, for instance of common salt, by holding it in contact with an excess of solid salt, we call the water the solvent, the salt which is dissolved in the water the solute, and the excess of solid salt needed to insure our saturating the water, the bodenkoerper, or solid phase. When the water has dissolved all the salt which it is capable of containing at this temperature as is shown by its leaving some solid salt unabsorbed we call the solution saturated, and this quantity of salt we call the solubility of salt in water at this temperature.

Admitting that the carbon in molten cast-iron exists as cementite, does the line *B'D'*, Fig. 1, represent the solubility of cementite in molten iron in this same sense? Manifestly not. The quantity of a salt which water will dissolve varies with the nature of the solid supply of that salt, the bodenkoerper, with which that water is in contact.

Roozeboom's[28] classical example of the varying quantity of thorium sulphate which water will contain in equilibrium with different bodenkoerpers is shown in Fig. 13. If water is in contact with the solid salt $Th(SO_4)_2, 9H_2O$ the quantity of $Th(SO_4)_2$ which 100 parts of it can contain at various temperatures is represented by the line *ABC*. Each of the other lines represents the quantity of $Th(SO_4)_2$ which water can contain when in contact with some one of the other solid hydrated thorium sulphates.

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[28] Ostwald, *Lehrbuch der Allgemeinen Chemie*, vol. i., p. 1062 (Leipsic, 1903).

Water in contact with a solid salt will dissolve that salt till equilibrium is reached, the equilibrium at which the passage of particles of the solute out of the solution to rejoin the bodenkoerper is just rapid enough to balance the simultaneous passage of particles from the bodenkoerper into the solution to join the solute. So two neighboring hot bodies are in equilibrium with each other when they are at the same temperature, *i.e.*, when passage of heat from A to B just balances the passage back from B to A. So water and steam are in equilibrium in the boiler when the rate of evaporation of the water just balances the simultaneous condensation of steam back upon the surface of that water. This conception of equilibrium as representing not stagnation but equal and opposite transfer is most important.

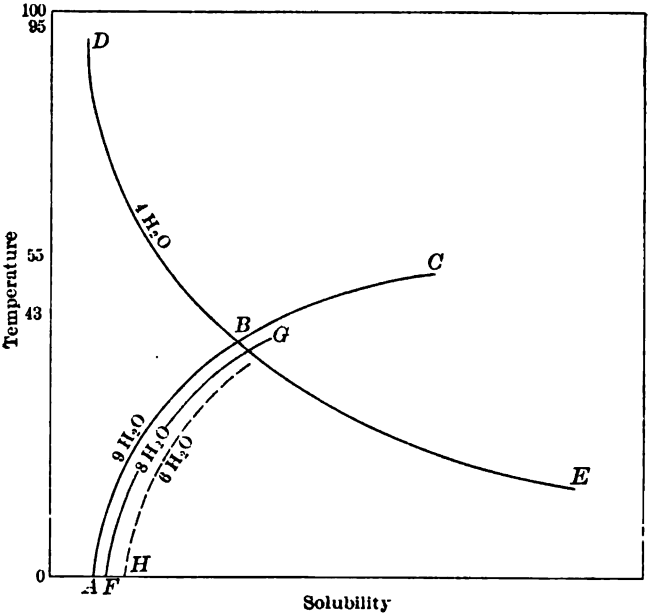

Fig. 13.—Solubility expressed as Parts of $Th(SO_4)_2$ per 100 Parts of Water. The quantity of Thorium Sulphate which water will dissolve at given temperatures varies with the nature (degree of hydration) of the Bodenkoerper or supply of solid Sulphate.

In the case of a solution we may conveniently say that when this equilibrium is reached the solution is "saturated." But the point to recognize clearly here is that the quantity or concentration of the solute in the solution at which this equilibrium is reached, *i.e.*, at which saturation occurs, varies with the nature of the bodenkoerper.

Turning now to the term "solubility," when we speak of the solubility of a salt in water we naturally have in mind the quantity of that salt which the water can contain when it is in equilibrium with an excess of that same salt in the solid state, *i.e.*, as bodenkoerper. We might indeed speak of the "solubility of salt in water when in contact with bodenkoerper A, B, or C;" but to prevent confusion it seems better to confine the term "solubility," (*SK*, *Saettigungskonzentration*) to the case in which solute and bodenkoerper are identical, and to use the term "equilibrium content" (*GK*, *Gleichgewichtskonzentration*) to represent the quantity of salt which the solvent can retain dissolved within it when in contact with any other bodenkoerper.

Philosophically speaking, solubility is only a single case of equilibrium content, representing the equilibrium when the solute and the bodenkoerper are identical (apart from ionization), and the distinction between true "solubility, *SK*," and "equilibrium, *GK*" is purely one of convenience.

In the experiments of Ruff and Goecke, and also in those of Hanemann, the molten iron was held in contact not with solid cementite, if indeed that is possible, but with solid graphite. The quantity of carbon which it took up, *B′D′*, therefore was not the true solubility of cementite in molten iron, *SK*, but the equilibrium content *GK* of carbon in molten iron in contact with graphite.

SUBJECT TO REVISION.

[TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS.]

The Wood Flotation Process.

BY HENRY E. WOOD, DENVER, COLO.

(Cleveland Meeting, October, 1912.)

In my opinion, the concentration of minerals by flotation is the most interesting problem in ore-dressing, and will command eventually far more consideration than it has at present. For many ores it furnishes a complete method of concentration absolutely independent of specific gravity, and thus simplifies the milling of many difficult combinations. Not having to recognize the influence of specific gravity, we are also able to dispense with some of the complications resulting from classifying-conditions.

Where flotation cannot be used for a complete separation of the minerals from the gangue, it will be found, in nearly all cases, to save those portions which are generally lost by reason of their fine state of division. For this reason, it should be looked upon mainly as an aid to general concentration, although there are many ores so well adapted to flotation that more than 90 per cent. of the valuable metals can be recovered from them in one operation by this process.

In presenting the results of my investigations in flotation, I have endeavored to theorize as little as possible. That side of the subject has been extensively discussed by others.

My earliest work with flotation was done in 1895, when I used various oils, grease, soap, etc., with results that were remarkable. I did not realize then that with water alone equally good separations could be made. Up to 1906, my endeavors were confined to attempts to keep the concentrates on the tables well wetted and away from exposure to the air. In common with others, I observed that when there was any obstruction in the wash-water on the table of sufficient size to divert the water to either side, the concentrates collected, and, on exposure to the air, would promptly float away. This convinced me that the sulphides were all good "swimmers," and that if, instead of endeavoring to drown them, they were given a chance, they would all float.

Having successfully demonstrated this fact, as well as another equally important, namely, that practically all the oxides are quickly wetted, become saturated like a sponge, and, sinking under the surface-film, become subject to gravity-conditions, I commenced my experiments.

I found that if dry-crushed ore, not necessarily very fine, was gently deposited upon a swiftly-moving sheet of water, separation of the sulphides from the gangue took place. Retarding the current permitted the gangue to sink, while a film of sulphides remained on the surface. To collect this film I have devised three types of machine, shown in Figs. 1, 2, and 3. Fig. 4 is a photographic view of the type *C* machine.

All three types are similar so far as the system of feed is concerned. In type *A*, a sectional view of which is presented in Fig. 1, the unsized ore, at whatever mesh has been decided upon, and ranging from 10- to 40-mesh or finer, is delivered to a feed-hopper, *A*, the outlet from which is under close control, so that a uniform amount of ore can be steadily dropped upon a plate, *B*, vibrating 500 times per minute. From this plate the ore falls direct to the sheet of water, which flows over a plane, *C*, inclined at 7°. The film of sulphides crosses the tank, *D*, and in falling over the outlet is dewatered and split off by means of a nearly vertical screen, *E*. O, O, are guides or stretchers, and perform quite an important function when this type of machine is used. The elastic properties of the film are such that if these guides were not used, the sheet of water emerging from the tank would quickly assume a spiral shape, which would result in a wetting of the sulphides. These would then pass through the screen instead of over it.

The guides are so adjusted that the water intersects the screen in a line parallel to the surface of the tank. By slight adjustments one is then able to split off cleanly the film of concentrates from the water and the gangue in suspension. A separate patent was allowed for this feature. The water passing through the screen, containing more or less fine gangue in suspension, is caught at *H* and conducted to a launder, *F*, which also collects such coarse mineral and gangue as steadily issues from the bottom outlet of the tank, *G*, from which it passes to tables. I

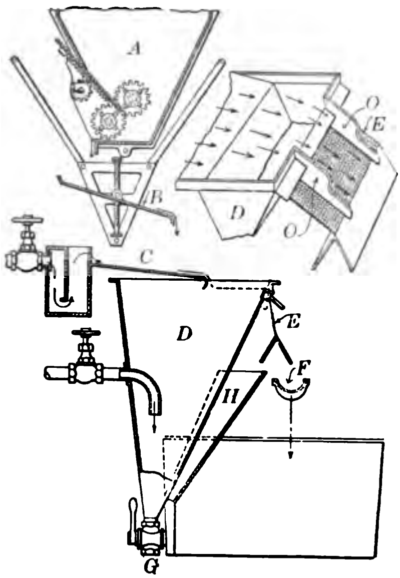

Fig. 1.—Type *A*.

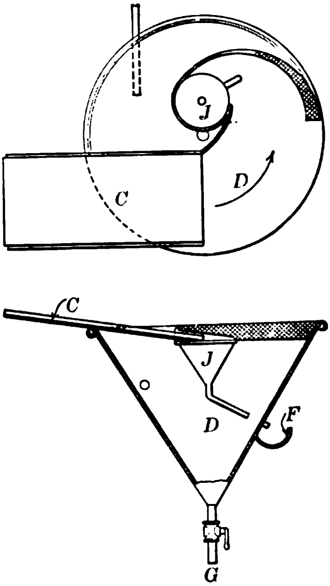

Fig. 2.—Type *B*.

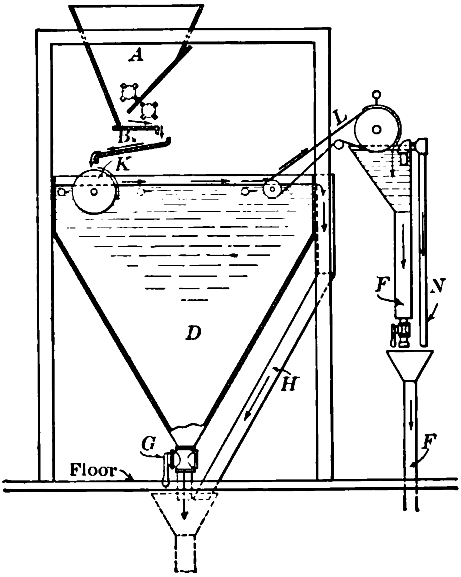

Fig. 3.—Type *C*.

FIGS. 1, 2, AND 3.—THREE TYPES OF THE WOOD ORE-CONCENTRATING MACHINE.

sometimes find it advantageous either to classify hydraulically or screen-size the tank contents *en route* to the table; but it is not essential in all cases, because, having removed the fine sulphides, as well as a large proportion of the larger sizes, by flotation, only the coarser mineral and the coarser gangue have to be considered, as the fine gangue in suspension passes off with the wash-water.

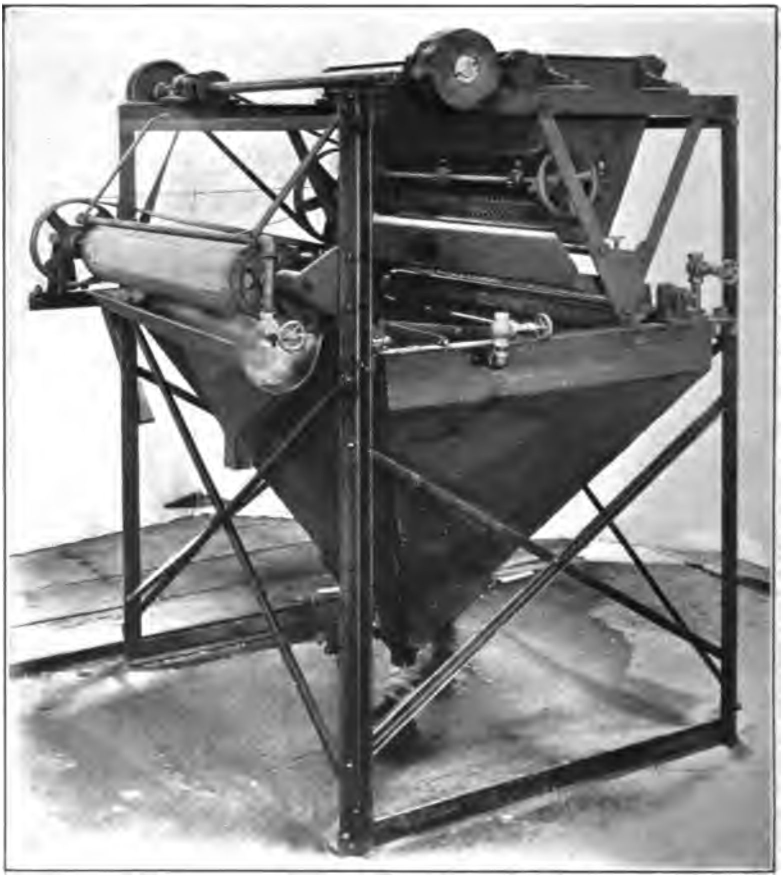

FIG. 4.—PHOTOGRAPHIC VIEW OF THE WOOD ORE-CONCENTRATING MACHINE, TYPE *C*.

The feed of the type *B* machine, Fig. 2, is delivered over the inclined plane *C* to a conical tank, *D*, in which a rapidly-revolving motion with vortical effect is given to the surface of the water by means of a water-jet against the side of the tank. The feed falls upon an inclined sheet of water. The film of sulphides is guided by means of partly-submerged vertical screens, *E*, to a funnel, *J*, the outlet of which penetrates the side of the tank, where the concentrates are delivered to a launder, *F*.

The output of a tank 5 ft. in diameter will exceed 0.5 ton per hour on a 20-mesh ore that is adapted to flotation. Necessarily, a small amount of water is required to carry the film into the funnel. The film, having a definite thickness, is probably carried on an amount of water of two or more times the volume of the concentrate. As this water carries more or less gangue in suspension, it makes the concentrates siliceous. In certain sections of Mexico, where copper sulphides are in demand and silica is not objectionable, the conical-tank plan meets the requirements of the situation; but this form would not be satisfactory, for instance, in Utah, where a high-grade copper-concentrate is of more importance than the highest possible recovery. In Utah there is a penalty imposed by the smelters for silica above a certain amount, a condition which does not exist in Mexico.

In the type *C* machine, Fig. 3, the 20- to 40-mesh ore is dropped from the vibrating plate, *B*, to a corrugated rubber- or canvas-covered cylinder, *K*, which is revolving in water. Enough water is elevated above the water-level of the tank to wet the gangue and float the sulphides. This sulphide film passes from the cylinder over the tank-surface to an endless canvas belt, *L*, which emerges from the tank and is controlled by sprockets. Water-jets gently force the concentrates upon the canvas, but, as the canvas is elevated, the water drains back to the tank, carrying with it any gangue in suspension. The concentrates are deposited in another tank and collected in a launder, *F*. An automatic siphon, *N*, prevents an overflow of concentrates from the tank after they are discharged from the belt. *G* is the outlet-valve controlling the discharge of tailings after flotation. In most cases the tailings pass to a Wilfley table. Quite frequently however, the recovery made by flotation makes any additional treatment unnecessary.

As clean wash-water is used at the delivery end of the take-off belt to remove the concentrates, they have a minimum silica-content. The difference between type *B* and type *C* in this respect was shown by comparative tests. The crude ore used in each test assayed 2.63 per cent. of copper. The *B* machine saved 88 per cent. and gave concentrates assaying 18 per cent. of copper, with 23 per cent. of SiO_2, while the *C* machine saved 93 per cent., giving concentrates assaying 27 per cent. of copper, with 3.6 per cent. of SiO_2, both tests being made by flotation alone.

As the corrugations of the rubber-covered cylinder are only adapted to comparatively fine ore, say 20-mesh and finer, they would not bring up enough water to saturate particles that were larger than the corrugations; hence, in treating 8- or 10-mesh ore, all of such particles would not reach the water, and other dry ore would fall on top of them, causing an unclean concentrate. To prevent such conditions in treating coarse ores, the cylinder is removed, and a rapid impetus is given to the surface of the water by means of water-jets from a pipe just in advance of the position formerly occupied by the cylinder.

An ore suitable for jigging, in which the sulphides are liberated at 4-mesh, can be fed, either sized or unsized, direct to the tank. The treatment of coarse ore, say from 4- to 12-mesh, is performed for the purpose of floating off fine particles of sulphides which mechanically adhere to the coarser sizes. I have carefully dry-screened ore from 4- to 20-mesh and then subjected these coarse sizes to flotation, the result being a high-grade product that would add 5 per cent. to the general recovery.

During my experimental work on more than 800 separate samples, many interesting incidents were observed, among which I note the following:

If, after grinding a given ore to the right size, it is at once flotated, from 30 to 85 per cent. by weight of its sulphide particles are likely to float. If another portion, prepared at the same time and under the same conditions, is set aside for several days and then treated, less than one-third of what floated before will remain on the surface. This applies particularly to the copper sulphides, and must be due to partial oxidation, although such a change is not visible to the eye. This behvior of copper-ores is more pronounced if steam-heat be applied in drying a previously-made concentrate for flotation. It is therefore not advisable to regrind wet jig-tailings and dry them for flotation. To secure the best possible flotation from copper-ores, they must be ground directly to the required mesh for separation of the mineral from the gangue, and promptly milled.

Curiously enough, this state of affairs does not exist in the treatment of some other complex sulphides, for instance, those containing galena and the non-magnetic resin blende. At 40-mesh a 10-per cent. lead-ore, with 20 per cent. of zinc, will produce a flotation-concentrate accounting for 80 per cent. of the lead, assaying from 56 to 60 per cent. of Pb and 7 per cent. of Zn. If the tailings, after flotation, are passed over a Wilfley table, a high-grade coarse lead-concentrate is easily cut out from the zinc and silica. The removal of the silica from the zinc-middlings enables us to obtain a high-grade zinc-concentrate. Now, if this blende, which sank in the tank during flotation, be dried and re-treated by flotation, nearly all of it will float, while the accompanying silica will sink, leaving a high-grade product. This particular effect is not so pronounced when the blende is of the magnetic or so called black-jack variety.

I have noted in the treatment of copper-ores that the tendency to float decreases as the sulphur-content diminishes. More than 90 per cent. of the copper of chalcopyrite ore with a hard white quartz gangue can be saved with less than 5 per cent. of silica in the concentrate by flotation alone; but if bornite or chalcocite is present, the proportions of these minerals saved are much smaller than of chalcopyrite. If chalcocite contains a small percentage of chalcopyrite, practically all of the latter will float and some of the fine chalcocite with it. This leaves the coarse chalcocite in good shape for final table-separation.

Most of the magnetic minerals will sink, so that a good flotation-concentrate can sometimes be made from a mixture of chalcopyrite associated with a magnetic blende. Magnetite will sink and leave suspended on the surface any fine sulphides that may be with it, as well as the metals, such as gold, silver, platinum, etc. To obtain the best effects for the fine metals, we sometimes modify the film by using a minute amount of oil. The oil being used purely as a conveyor, and not as an enveloping element, my patents do not conflict with the oil-systems. My main idea is the use of water alone. I rarely use oil and never acid or gas.

Judging from the appearance of a 20-mesh flotation-concentrate, the inexperienced observer assumes, from the smooth surface of the film, that only the extremely fine sulphides are being recovered. A screen-analysis of the concentrate, however, shows that more than 10 per cent. of the concentrate is between 20- and 40-mesh. Apparently the elasticity of the film is such that it supports the coarser particles, but not rigidly, so that only the apex of each piece of sulphide is shown, and as it lies along with the fines, a uniform, smooth surface is presented.

The strength of the film is illustrated in another way. If particles of dry ore are placed upon the film of sulphides, they will be thrown off by the rebound of the film as it passes over the curve of the dam, the effect being practically the same as the repulsion of highly-conductive minerals in passing over an electrostatically-charged roller.

The elastic properties of the film are plainly shown during the operation. The sulphide film spreads out rapidly over the tank-surface. If its passage is restricted, it will immediately be compressed into ridges, which in time become sufficiently heavy to sink of their own weight. Also, before sinking, the film will creep up vertically on the corrugated roller.

Violent agitation of the film must be avoided, or the particles will become wet and sink. Nevertheless, the tenacity of the film is remarkable.

As long as the feed is continuous, the sulphides on the film easily slip over to the take-off belt. This action is assisted by a gentle water-pressure directly in front of the belt. A gentle current of air also produces the same result.

I have alluded to oxidizing influences preventing or interfering with flotation-results in the treatment of copper sulphides. With several other sulphides this is not the case, so that flotation can frequently be applied, after amalgamation and concentration, as a more economical method of recovery than cyanidation of the tailings.

For instance, I had a $20 gold-ore, of which $15, or 75 per cent., was removed by amalgamation and concentration. Of the $5 left in the tailings, 93 per cent. was extracted by cyanide, but at a cost of $2.75, by reason of the high cyanide-consumption, caused by values still locked in the fine sulphides. The same tailings, assaying $5, were dried and flotated, producing a 10-oz. gold-concentrate and leaving 30 cents per ton in the final tailings. The cost was far less than that of cyanide treatment.

Nearly all the average free-gold ores of California contain arsenical pyrites, and this, in turn, carries gold that is not free. Such ores, if crushed dry and treated by flotation, yield high-grade concentrates, accounting for from 10 to 20 per cent. of the total valuable metals in the ore. After flotation, the free gold can be amalgamated as usual. I have found that all that is necessary is to feed over an amalgamated plate, which has an inclination of 7° and is covered by water.

Such flotation-concentrates of arsenical pyrites frequently assay many thousands of dollars per ton, and constitute an important item in the total recovery. A considerable percentage of the values in some varieties of wet-crushed ores, particularly of copper and arsenical sulphides in a hard gangue, will float. While we cannot do anything with sulphides in suspension, we can readily save all floating slimes from wet-crushing by this same general system, and with a simple device placed in any launder below the mill, provided the surface-conditions are not rough.

All over the world, valuable ores, such as sulphides of silver, gray copper, galena, etc., are frequently found in a matrix of baryta, or associated with heavy garnet. The specific gravity of such gangues practically prohibits either jigging or table-separation; but when this method of ore-dressing is applied to them, from 80 to 90 per cent. of their values can be recovered at one simple operation. A recent example of this was a 1,200-oz. silver-concentrate made from a Mexican ore containing 78 per cent. of $BaSO_4$. Another, from Canada—a crystallized barite containing 10 per cent. of Pb—gave a 90-per cent. recovery in a 70-per cent. lead-product. Many mines containing ores of such composition have been abandoned because of the closeness of the specific gravities.

We have had several ores of nearly clean pyrrhotite containing a small percentage of chalcopyrite. Most of the latter will float by itself. If the remainder is concentrated, the ratio of concentration is too low to make a valuable copper-product; but if this concentrate is dried and passed through a magnetic separator, such as the Wetherill or Dings, at a low amperage, the pyrrhotite is lifted off and the non-magnetic tailings will be a clean chalcopyrite. This, combined with the flotation-concentrate, gives a total high extraction.

The capacity of the series of flotation-machines here shown is variable, depending upon several conditions, among which are the nature of the gangue, the size of the ore, and the concentration-ratio. A standard machine treating a 20-mesh quartz ore, using a 3-ft. width of feed and having a 4-ft. take-off belt, will vary in capacity from 1,000 to 2,000 lb. per hour, unless the ratio of concentration is low, in which case the capacity will be smaller. Some ores that possess an easily wetted gangue and call for a high concentration-ratio can be fed rapidly at 20-, 30-, or 40-mesh.

For instance, a 1 or 2 per cent. molybdenite ore in a quartz gangue will give a clean concentrate, even if the ore is fed several times faster than an ordinary sulphide ore; and the same is true of graphite. In a special form of the machine, designed for the concentration of molybdenite and graphite, one or two partly-submerged cylinders, covered with either canvas or fine wire screen, are inserted in the tank. These cylinders revolve in the same direction, and at the same rate of speed as the main take-off belt or belts. The film-concentrate is lifted from the tank, and passes over the rollers down to the tank-level again. With each elevation partly-saturated gangue becomes thoroughly wetted, and drops off into the tank without any apparent disturbance or wetting of the concentrate. Such action naturally raises the grade of the concentrate. In leaving the take-off belt, it is not washed off by direct water-jets, as is done with other ores, but the take-off belt is allowed to discharge its load at water-level in a shallow compartment, and the concentrate gently floats off to a collecting-launder.

In the flotation-treatment of molybdenite it frequently happens that copper sulphides, iron pyrite, and other impurities, are present, and must be removed to make a salable product. If copper and iron are present, they are re-concentrated on a Wilfley table, so handled that the tailings will contain the molybdenite. Another way is to dry the flotation-concentrate, give it a gentle roast, and pass it through a magnetic separator, to remove the copper and iron. Molybdenite- and graphite-concentrates are frequently contaminated by mica. This in some cases can also be taken out by the magnetic separator, or by re-flotation with the water at 110° F. It can also be removed electrostatically.

Objections are made to the drying of flotation-concentrates on account of their supposed fineness, but this is a mistake, as mentioned previously. There should be no more difficulty in drying concentrates of this nature than those produced by vanners or ordinary slime-tables. By alternating settling-tanks, which can be siphoned, or by using filter-presses, they can be constantly dewatered. They can be briquetted if necessary.

There are, in Mexico and elsewhere, dry-crushing mills that use magnetic separators to remove large amounts of gangue in the shape of garnets, siderite, etc., in order to get more favorable conditions for gravity-separations on the tables for the lead and zinc. If a flotation-machine be placed so that it will take the non-magnetic tailings before they reach the table, a high-grade lead-product results, and aids in the general separation of the coarse lead and zinc, improving each from 15 to 20 per cent.

Flotation will be found to be of the greatest benefit when applied to ores that have the largest losses in milling, as well as to certain ores that are virtually not subject to wet concentration on account of small differences in specific gravity.

The proper preparation of ores for flotation is essential. For instance, if a small sample is pulverized unnecessarily fine, in such a state as would result from the use of an ordinary muller, poor returns may be expected, but if the same ore is shattered by pounding on an iron plate, most favorable products will ensue, as the crystals retain more of their original shape.

For this reason, I have found that ores should be prepared in ball-mills, or by rolls, rather than in any machine that grinds much of the gangue to an impalpable dust, in which condition both the gangue and the mineral of many ores will float. Flotation-treatment of definitely-sized ore will materially increase the capacity of the large sizes by themselves, but mechanical difficulties are encountered in feeding the smaller sizes by themselves, and the capacity is also much reduced; hence it is advantageous, after having decided upon the proper mesh to liberate the sulphides from the gangue, to feed them all at once without previous sizing.

I wish, also, to place special emphasis upon the property of easy saturation or wetting of nearly all of the oxides. The only one that has surprised us and floated in appreciable proportions is the specular hematite in certain New Mexico zinc-ores, which floats up to 10 or 20 per cent. by weight. This is largely due to the tabular shape of its crystals. After magnetic removal of the iron, the non-magnetic tailings assay from 45 to 50 per cent. of zinc.

Simple as these flotation-devices are, it has taken several years to secure patents, both domestic and foreign, as well as to develop the general standard type of machine now on the market. It weighs 1,000 lb. net; measures 4 by 5 by 6 ft. Ready for mule-back transportation, its weight is 1,100 lb. It requires 0.25 h-p. and less water than is used in wet concentration.

Sulphides of the following metals are particularly favorable for flotation methods: Copper, lead, silver, antimony, bismuth, mercury, arsenic, molybdenum, zinc, iron, nickel, etc., as well as tellurides, graphite, gold, and platinum.

When tellurides are present, a high-grade flotation-concentrate is obtained and the tailings are left in excellent shape for cyanidation. I have proved this commercially on ores from Boulder county, Colo.

RESULTS ON MOLYBDENITE.

The concentration of molybdenite by flotation is decidedly interesting, in that it furnishes illustrations of several of the peculiarities of surface-tension. For instance, in the recent flotation of a Canadian ore of this nature, with a quartz-schist gangue, which assayed 1.87 per cent. of MoS_2, I obtained an apparently pure film of MoS_2, which, when assayed, was found to contain only 45.31 per cent. of MoS_2, and 41.97 per cent. of insoluble residue. On examination, the under side of the film was seen to be composed of fine mica scales, which must have adhered to the flakes of MoS_2. My first conclusion, that the mica was simply enveloped in a coating of MoS_2, was not correct. The recovery in this particular case was 79.08 per cent. The test was made with water having a temperature of 56° F.

The above examination was followed by that of an 8.65-per cent. molybdenite ore from Alaska. This was a complex ore, containing pyrrhotite and magnetite, and low in silica. The flotation-concentrate, at 40-mesh, amounted to 23.4 per cent. by weight. As it assayed only 30 per cent. of MoS_2, a recovery of 79 per cent., it was evidently contaminated by other minerals that had floated.

Another sample of the same ore, lightly roasted, gave a flotation-concentrate assaying 81.45 per cent. of MoS_2, the ratio of concentration being 35.71 : 1, but the saving was only 26.3 per cent. This indicates that either some of the molybdenum is present in the original ore as an oxide, or that the roasting was carried too far and oxidized it. In the first test the saving was 79 per cent., assaying 30 per cent. of MoS_2, while the second trial recovered but 26 per cent., assaying 81.45 per cent. It is possible, therefore, that by obtaining the right roasting-conditions, such an ore could be profitably treated.

The concentrates from one test of this ore assayed 1.86 per cent. of copper. After a gentle roast, this was removed by the Wetherill separator, leaving only a trace of copper, and with no material loss of MoS_2.

A certain Alaskan ore recently treated by me in commercial quantities furnishes an illustration of the extreme sensitiveness of chalcopyrite to oxidizing influences. It contained 4 per cent. of copper, closely associated with a heavy pyrrhotite in a siliceous gangue. Careful work on this ore by jigging, tabling, regrinding of jig-tailings, all followed by canvas slime-table treatment, gave a 15 per cent. copper-concentrate, with a recovery of 62 per cent. Direct table-concentration made a 16 per cent. copper-concentrate, saving 47 per cent. The same ore, by flotation alone, made a 25-per cent. copper-concentrate, accounting for 69 per cent. of the copper. To this was added 11 per cent. from the table-work, a total of 80 per cent. In making this examination the ore was dry-crushed at 8-mesh, and the from 8- to 20-mesh product (dry-screened) was passed through the flotation-machine, resulting in a high-grade concentrate, which added 4.25 per cent. to the total recovery. This proportion came from the mechanically adhering fine sulphides, and would be lost in ordinary jig-work. The from 8- to 20-mesh jig-tailings of this test were dried and reground for flotation; but on account of the oxidation not a particle floated.

Screen-Analysis of 20-Mesh Flotation-Concentrates.

	Mesh.	Percentage by Weight.
No. 1	20 to 40	14.00
No. 2	20 to 40	10.00
No. 3	20 to 40	41.75

Many similar examples could be cited showing that flotation largely reduces the proportions going to the tables, thereby increasing the efficiency of the latter.

Table I. gives the results obtained by flotation alone. All of the ores tried were difficult ones to treat, and several of them were not susceptible to gravity-separation. The recovery by flotation alone, in many instances, exceeded the returns from the best possible treatment in the wet way.

TABLE I.—*Results Obtained by Flotation.*

Locality.	Mesh Size.	Assay of Ore.	Concentrations.	Assay of Flotation-Concentrates.	Recovery by Flotation (Concentrates).	Total Recovery.
Copper-Ores.		Per Cent.	Tons Into 1.	Per Cent.	Per Cent.	Per Cent.
Canada. Chalcopyrite	40	Cu. 2.60	8.25	17.10	79.72	94.48
Mexico. Chalcopyrite	40	Cu. 5.80	8.28	13.20	69.40	88.45
Colorado. Chalcopyrite	40	Cu. 1.42	5.97	5.40	68.75	83.29
Colorado. Chalcopyrite	40	Cu. 2.60	4.75	12.25	96.28	98.45
Canada. Chalcopyrite	40	Cu. 2.75	9.38	13.45	52.15	86.05
Colorado. Chalcopyrite	40	Cu. 2.50	7.04	15.00	71.59	88.35
Nevada. Chalcopyrite and garnet	40	Cu. 3.85	9.80	15.45	40.94	91.21
Arizona tailings	40	Cu. 0.47	23.25	4.80	43.95	74.16
Mexico	40	Cu. 15.20	2.14	26.50	81.48	89.01
Montana	40	Cu. 1.90	....	11.40	42.92	90.45
Colorado	40	Cu. 8.28	....	21.60	52.71	91.57
Gold-Ores.		Oz. Per Ton.		Oz. Per Ton.		
Colorado	40	Au 0.26	26.66	6.32	91.19	91.19
Mexico	40	Au. 0.42	5.83	1.00	40.98	94.41
Colorado	40	Au. 0.66	5.97	1.42	36.04	86.48
Colorado	40	Au. 3.50	4.75	4.86	29.24	89.62
California. Arsenical pyrite	40	Au. 90.96	14.59	311.00	23.43	94.52
Arizona	40	Au. 0.12	23.25	0.98	35.18	80.12
Silver-Ores.						
New Mexico	40	Ag. 15.90	23.25	184.90	50.01	90.01
Mexico	40	Ag. 95.60	5.88	340.90	61.11	86.26
Colorado	40	Ag. 3.00	5.97	10.60	59.12	88.61
Colorado	40	Ag. 12.96	4.75	43.30	70.34	84.65
Lead-Ores.		Per Cent.		Per Cent.		
Colorado	40	Pb. 12.20	11.11	54.00	39.85	90.86
Colorado. Baryta	40	Pb. 16.50	6.06	41.60	41.60	95.82
Canada	40	Pb. 19.70	5.35	59.60	56.55	79.44
Mexico	40	Pb. 5.60	5.83	16.90	51.77	89.65

Copper-Ores.

Alaska.—30 mesh, 4.08 per cent. of copper. Close classification, jigging, table-concentration, with regrinding of jig-tailings, gave 65 per cent. of the copper, assaying 15.66 per cent. of Cu. Ratio, 5.9 tons to 1.

By flotation at 30-mesh—the same ore—the concentrate assayed 25.03 per cent. of Cu, recovering 73 per cent. at a ratio of 8.4 into 1. The Wilfley concentrate added to this 7.31 per cent. more, assaying 6.32 per cent. of Cu, ratio 21.19 to 1, a total recovery of 80.31 per cent. When treated at 40-mesh by flotation alone, the recovery exceeds the above total.

Colorado.—40-mesh.

	Copper-Assay. Per Cent.	Recovery. Per Cent.
Crude, 100 per cent. by weight	2.68	
Concentrates, 4.75 tons to 1	12.25	96.23

Silver-Ore.

South America.—30-mesh.

	Assay. Oz. Per Ton.	Recovery. Per Cent.
Contents	36.20	
Flotation-concentrates, ratio 3.64 to 1	106.90	80.90
Wilfley concentrates, ratio 16.01 to 1	36.00	6.19
Total at 2.97 to 1, assaying (average)	93.70	87.09

A second test of this ore gave 91 per cent. of its gold-content, 96.3 per cent. of the silver, 93.9 per cent. of the lead, and 84.41 per cent. of the copper. Several other combination methods applied to this ore failed to produce satisfactory commercial products.

Mexico.—20-mesh.

	Assays.		Recoveries.	
	Silver. Oz. Per Ton.	Lead. Per Cent.	Silver. Per Cent.	Lead. Per Cent.
Crude	36.5	9.00		
Flotation-concentrates, 17.4 per cent.	174.70	40.75	83.40	79.00
Wilfley concentrates, 8.34 per cent.	27.90	7.80	6.36	7.25
Total			89.76	86.25

The ore from Mexico when treated most carefully by wet-concentration gave 75.22 per cent. of the silver and 74.38 per cent. of the lead, both silver and lead being of much lower grade.

Mexico.

	Silver-Assay. Oz. Per Ton.	Recovery. Per Cent.
Silver sulphide in quartz	38.2	
Direct table-concentration at 20-mesh, ratio 20.97 tons to 1	395.6	49.38
The same ore treated by flotation alone—9.09 tons into 1	323.0	93.02

Bismuth-Ore.

Colorado.

	Bi. Per Cent.	Recovery. Per Cent.
Crude, 100 per cent. by weight at 40-mesh	1.00	
Flotation-concentrates, 3.84 per cent. by weight	21.45	55.52
Wilfley concentrates, 9.34 per cent. by weight	3.25	34.77
Total concentrates, 7.52 tons into 1	6.79	90.29

Gold-Ore.

Nevada.

	Assays.		Recovery.	
	Au. Oz. Per Ton.	Ag. Oz. Per Ton.	Au. Per Cent.	Ag. Per Cent.
At 20-mesh, 100 per cent. by weight	5.52	6.50		
Flotation-concentrates, 1.15 per cent. by weight	73.76	122.80	15.36	21.72
Wilfley concentrates, 7.90 per cent. by weight	43.26	48.20	61.96	58.62
Total concentrates at 11.04 to 1	48.87	57.58	77.31	80.34

Left in the tailings—$25 in gold and 69 cents in silver, in excellent condition for cyanide treatment, the sulphides having been removed.

Telluride Ore.

Colorado.

	Assays.		Recovery.	
	Au. Oz. Per Ton.	Ag. Oz. Per Ton.	Au. Per Cent.	Ag. Per Cent.
Crude ore	2.10	3.60		
Flotation-concentrates, 2.2 per cent. by weight	36.50	52.40	37.24	32.14
Wilfley concentrates, 6.84 per cent. by weight	11.14	12.70	36.33	24.63
Total concentrates, 11.06 tons into 1	17.31	22.36	73.57	56.77

With extraction of the tellurides the values remaining in the tailings were successfully cyanided.

Zinc-Ore.

Utah.

	Zn. Per Cent.	Recovery. Per Cent.
Slimes	24.50	
Flotation-concentrates, 3.59 tons into 1	48.20	59.67

Antimony-Ore (Stibnite).

	Sb. Per Cent.	Recovery. Per Cent.
Crude, 100 per cent. by weight at 40-mesh	3.57	
Flotation-concentrates, 6.50 per cent. by weight	47.40	86.30

Mercury-Ore (Cinnabar).

	By Weight. Per Cent.	Hg. Per Cent.	Per Cent.
Crude	100.00	6.30	
Flotation-concentrates	9.33	28.10	Recovery 41.58
Wilfley concentrates	4.33	76.90	Recovery 52.75
Tailings	86.33	0.40	Contain 5.48
Assay of combined concentrates, 7.32 tons into 1		43.56	Recovery 94.35

Molybdenite Ore.

Colorado.

	MoS_2. Per Cent.	Recovery. Per Cent.
Crude, 40-mesh	2.08	
Flotation-concentrates, 5 per cent. by weight	41.38	99

Re-treatment would materially increase the grade with little loss of MoS_2.

Alaska.

	MoS_2 Per Cent.	Recovery. Per Cent.
Crude, 20-mesh	2.00	
Flotation-concentrates, 3.33 per cent. by weight	61.55	92.27

Canada.

	MoS_2 Per Cent.	Recovery. Per Cent.
Quartz and mica gangue	1.87	
1st Test, H_2O at 56° F. Flotation-concentrates, 30.64 tons to 1	45.31	79.08
2d Test, H_2O at 110° F. Flotation-concentrates, 29.41 tons to 1	56.70	89.17

Foreign.

	Mesh.	MoS_2 Per Cent.		By Weight. Per Cent.	MoS_2 Per Cent.	Recovery. Per Cent.
(1)	20	6.66	Flotation-concentrates	10.5	63.50	96.34
(2)	30	6.73	Flotation-concentrates	8.6	67.42	86.26
(3)	40	6.66	Flotation-concentrates	8.8	65.50	86.48
(4)	40	8.95	Flotation-concentrates	11.29	74.60	94.19

Graphite.

Canada.

	Mesh.	Carbon. Per Cent.		By Weight. Per Cent.	Carbon. Per Cent.	Recovery. Per Cent.
(1)	40	19.46	Flotation-concentrates	25.55	69.38	91.27
(2)	40	65.34	Flotation-concentrates	72.00	79.92	88.12
(3)	40	6.30	Flotation-concentrates	6.80	71.14	76.79

Silver-Ore—Gray Copper—Galena—Baryta.

Colorado.

		Assays.		Recovery.	
	By Weight. Per Cent.	Silver. Ounces.	Lead. Per Cent.	Silver. Per Cent.	Lead. Per Cent.
Crude, 20-mesh	......	5.2	13.2	......	......
Flotation-concentrates	18.4	21.5	58.3	76.2	81.3
Wilfley-concentrates	6.9	10.0	20.7	13.3	10.8
	25.3	18.4	48.0	89.5	92.1

Blast-Furnace Flue-Dust.

Pennsylvania.

	By Weight. Per Cent.	Iron. Per Cent.
Flotation-concentrates (coke)	11.8	......
Tailings (iron)	85.0	64.60
Loss	3.2	......
	100.00	

SUBJECT TO REVISION.

[TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS.]

Notes on Titanium and on the Cleansing Effect of Titanium on Cast-Iron.

BY BRADLEY STOUGHTON, NEW YORK, N. Y.

(Cleveland Meeting, October, 1912.)

[SECRETARY'S NOTE.—To avoid repetition of foot-notes, references to authorities are made in this paper by means of figures, referring to a numbered list in the appendix.—J. S.]

INTRODUCTION.

THIS paper gives the results: first, of a research into the somewhat extensive literature of titanium; and second, of a series of original tests upon the effect of titanium on iron castings. New investigations have also been made upon the effect of titanium on steel, but I hope to treat this subject in an independent paper at some later date, and, therefore, will not discuss the experiments in detail here.

The combined results of this study, which extended over a period of more than two years, can be briefly summarized as follows:

1. Steel or cast-iron, in which the titanium is properly proportioned, and which has been properly treated, is improved in strength, toughness, and durability against wear, such as, for example, the wear of railroad-rails, of steel and chilled-iron rolls, car-wheels, etc.

2. These improvements seem to be caused, not so much by the direct effect of titanium on the metal, as by its cleansing influence in removing harmful impurities, such as oxygen and oxides, nitrogen, occluded slags, and perhaps also sulphur. It appears also to reduce segregation, which would contribute to the same end.

3. In order that its effects may be fully realized, the treatment of steel or iron with titanium must be correctly performed. There are a few simple, but essential, details to be observed; if they are neglected, the best results cannot be expected.

To illustrate by an example: A series of tests on cast-iron was carried on in a German foundry, in which scientific apparatus and instruments were employed, and great care apparently exercised in executing the work, but the titanium treatment was performed without a knowledge of the conditions which should have prevailed, and, as a consequence, the results were inconclusive. It is as if one should attempt to boil water with the best adapted appliances obtainable; with scientific instruments to measure the temperature and regulate the operations, and with every condition fulfilled except heating the water to a high enough temperature. Obviously, the result would be quite as unsatisfactory at 211° F., as far as boiling water is concerned, as at 60°, although the former point is within a hair's breadth of success. So, in adding titanium to iron or steel, if it is put into the slag, instead of the metal, or if opportunity is not given for it to be absorbed and do its work, or if the steel is subsequently allowed to become oxidized, or if the treated metal is cast much too hot or much too cold—if any, or all, of these important details are neglected, we must not expect to get the best results.

HISTORY OF TITANIUM.

Although the existence of the element which we now know as titanium was discovered and reported by McGregor[4A] in 1791, its real scientific record did not begin until 1794 and 1795, when a mineral specimen of rutile and two specimens of cyano-nitride of titanium were studied by Professor Klaproth,[5] and by Vauquelin,[7] Hecht,[7] and Hauey.[8] Professor Klaproth—the same who gave the metal uranium its name—called the new element "Titanium—in honor of the Titans, the first children of the earth."[6—p. 9] The first attempt to isolate it was made by Lampadius,[9, 11] who, in 1797, attacked a synthetic oxide by charcoal and heat, and obtained crystals of a dark copper color which he believed to be the metal, but which we know from repetitions of the experiment, and from the color of the product, to have been the cyanide or nitride, or both. In 1825, Berzelius[15] reduced the fluo-titanate of potassium—K_2TiF_6—by means of metallic potassium and heat, and obtained an amorphous gray powder, which is stated by Woehler[16—p. 181] to be the first specimen of metallic titanium produced. Subsequently, Woehler[16] himself, and other investigators who preceded him, obtained the metal by reduction with potassium, sodium, electrolysis, and other means; in each case the product was a black or grayish-black powder, or silver-white grains.[23] Notwithstanding these researches, the cyano-nitride of titanium, with its characteristic red color—said to be the only substance in nature resembling copper in appearance—was commonly believed to be metallic titanium until the publication of Woehler's classic monograph[16] in 1849. This paper of Woehler's contained the first published analysis of the copper-colored mineral, which had been originally studied in 1795, as mentioned above, and which had long been familiar to many iron-masters through its presence as accretions in blast-furnaces and as occluded crystals in slags.

So important had titanium become among the metals in 1864 that Rivot[1] devoted a chapter of his *Docimasie* to it, and, in 1879, Hallock[2] published a bibliography on titanium comprising several hundred references. In 1886 Shimer[33] further advanced our knowledge by showing that the traces of titanium rarely found in pig-iron occurred in the form of the carbide—TiC—at least in the five or six samples which he examined, and T. W. Hogg[37] extended this in 1893 by his investigations of ferro-manganese, in which he found millions of crystals of titanium cyano-nitride, with occasionally the nitride and carbide.

Rossi[3, 63, 110 A] has repeatedly shown that titaniferous iron-ores can be smelted successfully in the blast-furnace, but the presence of titanium compounds has nevertheless always been considered by most smelters a serious handicap. This phase of the subject was discussed by Riley[17] in 1863, Bowron[108 A] in 1882, and the first of Rossi's papers[3] appeared in 1890. This question forms a most interesting study by itself, but of late years has been somewhat superseded in the literature by accounts of the manufacture of ferro-titanium in the electric furnace. Before we come to that process, however, it is appropriate to mention the first production of melted specimens of metallic titanium, which was accomplished as a laboratory-experiment by Moissan[18] in his electric furnace in 1894.

Previous to 1895, Auguste J. Rossi[110] appears to have produced ferro-titanium, by an industrially practicable method. Its possibilities were, however, very slow in being recognized.

What may be regarded as the first extended application to steel began at the Maryland Steel Works in November, 1907; to cast-iron,[91] at the car-wheel foundry of the Norfolk & Western railway at Roanoke, Va., in August, 1908. The rapidity of progress since that time may be learned from the statistics of the subsequent years' production: In 1911, in America alone 79 manufacturers of steel treated a total of 410,600 tons of steel with ferro-titanium alloy—the average being about 9 lb. of alloy (containing from 10 to 15 per cent. of titanium) per gross ton of steel. A large, but unrecorded, amount of cast-iron was also treated. The present importance of this treatment will be indicated by comparison with other steel-alloying elements; thus: more steel and more cast-iron was treated with titanium in 1911 than was treated with all the other elements combined, except manganese and silicon (aluminum).

OCCURRENCE.

Titanium has often been spoken of as one of the "rare elements," but this description is true only in the sense that the metal is seldom met with in nature and in the arts under its own name and identity. This is because it unites with extraordinary eagerness with both the oxygen and the nitrogen of the atmosphere and with other elements and metals prevalent in nature, and because it can be separated from these combinations with much difficulty, so that it has never been found in the free state, and has only rarely been reduced from its compounds by artificial means. In every other aspect it is far from being rare: a reliable estimate places it among the ten commonest elements in the earth's crust, together with oxygen, aluminum, iron, silicon, etc.[45 A] It has often been recognized in plants, in the human blood, in meteorites, in the spectrum of the sun,[29] and elsewhere. It is frequently found in clays, where it has the effect of increasing their fusibility.[45] Numerous references tell of its usefulness in the arts for a hundred years back—long before electric smelting made it commercially available in metallurgy. Indeed, the widespread prevalence of titanium prevented giving herewith, as I had hoped to do, a comprehensive bibliography of the literature relating to it, because the publication in our *Transactions* of far more than a thousand references on the metallurgy and chemistry of titanium alone would scarcely be warranted. Even a list of references to the occurrence of titanium in connection with iron-ores would cover a period beginning in the eighteenth century,[10], [8] and would far exceed the space available here. Washington[46] reports that titanium is probably never wholly absent from any rocks, but that it is distinctly more abundant in basic than in acid rocks, and its affinities lie more with iron and sodium than with magnesium and, still less, with calcium; indeed, rocks high in calcium and phosphorus seldom contain much titanium. This has often been noted in iron-ores, which are more free from phosphorus when associated with titanium.

From the standpoint of the blast-furnace manager, iron-ores are not classed as "titaniferous" unless they contain more than 2 or 3 per cent. of titanium, because these small amounts give no especial difficulty in smelting; but vast deposits—even mountains—of iron-ore lie idle in various parts of the world[42, 49, 3 (pp. 93–4), 2] because they contain from 5 to 50 per cent. and more of titanium oxide or of the so-called "titanic iron"—a mineral whose formula is stated by Koenig and von der Pforten[36] to be $FeTiO_3$. Most of these titaniferous ores are magnetic, hematite being only infrequently accompanied by titanium.[43] The best known deposits are in Scandinavia,[40], [44], [46] Canada,[32] New York State,[39], [30] Lake Superior,[10] Australia,[27] and Wales.[41]

In the smelting of ores containing titanium, the latter usually passes more or less completely into the slag in oxidized form.[57] However, it does not require a higher temperature than is often employed in the iron blast-furnace to reduce oxide of titanium to the nitride, cyanide, or carbide, [9], [13], [14], [16] and accretions of the cyanide and nitride are often found in the hearth, as well as occasionally at higher levels, its presence in the latter location being due, no doubt, to the volatility of the cyano-nitride and its condensation in cooler regions.[66, 69 A, 79 (p. 482)] Microscopic crystals of cyanide and nitride are often seen in blast-furnace slags[22], [23], [31] and have been found in ferro-manganese, as already mentioned.[37] I am indebted to J. E. Johnson, Jr., for the following explanation (in a private communication of July 18, 1912) why cyano-nitride is more frequently found in ferro-manganese than in pig-iron:

> "Titanium is a highly refractory substance, and in small quantities is normally slagged out of the furnace. In order to make spiegel, and particularly ferro-manganese, it is necessary to run the furnace very hot and also very basic to prevent scouring of the manganese into the slag. I should say that these were the two conditions best calculated to reduce the rutile to cyano-nitride."

As regards the presence of metallic titanium in the elemental condition either in blast-furnace slag or in pig-iron, I have been unable to find any evidence to indicate that it occurs as such. Wollaston, in his oft-quoted treatise,[22] speaks of "metallic titanium found in blast-furnace slag," but his paper shows that he mistook the cyano-nitride for the metal, as most investigators did previous to 1849. De Greswolde,[23] who followed Wollaston, and in 1829 dissolved a titaniferous blast-furnace slag in aqua regia, did indeed obtain black and silver-white metallic powder and grains, but Klaproth,[5], [6] had already shown in 1795 that both nitric and hydrochloric acids would dissolve the cyano-nitride, which may therefore have been the origin of any metallic titanium fortuitously produced by de Greswolde, if, indeed, his grains were titanium. The methods of analysis used on pig-iron would not distinguish between titanium in the elemental state, or as cyanide, nitride, or carbide, and the many occurrences of titanium in pig-iron reported from the first half of the nineteenth century onward, indicate nothing more than that titanium was present in some form or other. Even as early as 1841 Karsten[51—p. 534] expresses the opinion that the titanium present was probably mechanically mixed, and not combined with the iron. Wedding[61—Vol. I., p. 373] expresses a like opinion. Percy,[54] who quotes several analyses, says that it appears to be present as TiO_2, while Bell[58—pp. 157, 167] mentions the presence of "titanium or its nitride." In view of the irreducibility of compounds of titanium by carbon at temperatures higher than those of the blast-furnace, even in the presence of iron, I am inclined to agree with Rossi that the small amount of titanium found—only rarely exceeding 0.50 per cent. and somewhat doubtful when over 1 per cent.[81]—was always in the combined state.

PHYSICAL PROPERTIES OF TITANIUM AND ITS COMPOUNDS.

Weiss and Kaiser[4—p. 392] determined the melting-point of a mass of impure titanium containing Ti, 95.1; Al, 2.78; H, 1.07; C, 0.42; undetermined, 0.33 per cent., and report it somewhat vaguely as between 2,200° and 2,400° C. (3,992° and 4,352° F.). This figure has often been quoted as the melting-point of the element, but, obviously, the presence of the impurities mentioned would alter the fusing-point materially. Both Moissan and Rossi, who have probably had more experience with the metal in molten masses than any other scientific investigators, give much higher figures. Moissan says[18] that titanium with 2 per cent. of carbon is the most refractory substance he has met with in his electric-furnace experiments, and Rossi[110 G] affirms that the melting-point of titanium is between 2,700° and 3,000° C. (4,892° to 5,432° F.).

However, Hunter[116] contradicts both these estimates, and explains them on the ground that Moissan and others produced, not pure metal, but alloys of the metal with its compounds. His estimate of the melting-point is between 1,800° and 1,850° C.

Weiss and Kaiser give the following figures for constants of what they call their "pure metal"[4—p. 395]—*i. e.*, Ti, 97.41; H, 1.56; C, 0.30; undetermined, 0.73 per cent.; specific gravity, 5.174; specific heat, 0.1418; atomic heat, 6.830; heat of combination with oxygen: $Ti + 2O = TiO_2 + 97{,}772$ calories. I much favor, however, the figures for specific gravity given by Moissan, Rossi, and Hunter; the first-named gives 4.87;[18] the second, 4.85[110 G] and 4.84, and, finally, Hunter says 4.50.[116—pp. 335-6] The color of the metal in massive form is silver-white, with a brilliant white fracture.[4, 18] It is hard and brittle when cold, but can readily be forged at red heat.[116—pp. 335-6] The melting-point of ferro-titanium, made by the thermit process, and containing from 20 to 25 per cent. of titanium, is said to be between 1,300° and 1,400° C. (2,372° and 2,552° F.).[119 A]

Titanium seems to give extreme hardness to some of its alloys and compounds; thus, its ternary alloy with aluminum and copper is said to be very hard and resilient; the cyanides, nitrides, and carbides are very hard, indeed. The cyano-nitride, according to Vauquelin and Hecht,[7] is hard enough to scratch glass, but is pulverized with difficulty.[7—p. 11] According to Rivot,[1—p. 131] it will even scratch quartz, but, on the contrary, he says that it is very fragile, and is pulverized with ease in a mortar. Lampadius[11] found it brittle, but possessing in small scales a considerable degree of elasticity. These differences, as well as others to be noted, are no doubt explainable, in part at least, by the variable composition of the cyano-nitride, which fluctuates greatly in the relative proportions of cyanide and nitride contained; for example, the specimen examined by Woehler,[16] whose analysis has been so often quoted, contained Ti, 77.26; N, 18.30; C, 3.64 per cent., while the crystals separated by Hogg from ferro-manganese [37] contained from 60.5 to 79.8 per cent. of Ti, and always some iron. The cyano-nitride has a brilliant metallic luster, and a markedly characteristic red-brown, or copper, color. It crystallizes often in cubes, sometimes as small as 0.001 in. in diameter; less often in octahedra and other isometric forms; it is not easily fused,[11, 67] but volatilizes at the temperature of the crucible-steel melting-furnace [66, 69 A, 79] (which would be about 1,750° C.). Its specific gravity is given by different observers as follows:

1796, Vauquelin and Hecht, reference No. 7, specific gravity.....4.2469
1822, Wollaston, reference No. 22, specific gravity.....5.3
1897, Franck, reference No. 67, specific gravity.....5.28
1895, Hogg, reference No. 37, specific gravity.....4.1 to 5.1

Shimer gives the specific gravity of the carbide—TiC—as 5.1[33]. Moissan[18A] gives it as 4.3. Wollaston[22] says that the electric conductivity of the cyano-nitride is great.

Titanium has the power of forming compounds having a variety of colors of unusual richness, such as red, green, yellow, blue, etc. For instance, the different nitrides studied by Woehler were violet, golden yellow, and copper colored. In this property titanium resembles gold. The compounds of titanium also vary greatly in their state of aggregation—from the extraordinary infusibility of the cyano-nitride, to the fluidity of the chloride, and the volatility of the chloride [68] and hydrofluoride.

CHEMICAL PROPERTIES OF TITANIUM.

The chemical properties of titanium are, in general, similar to those of silicon, with the difference that titanium is more active in combination, both as to the number of elements with which it unites readily, and the number of different compounds formed with each. In some respects it is also far more energetic in its action than silicon; as, for example, it unites with oxygen, nitrogen, chlorine, and bromine with so much energy as to become incandescent.[16 (pp. 181-2), 18, 73 (p. 126)]

It unites readily with oxygen at 610° C. (1,130° F.) and this reaction, according to Kaiser,[4] evolves 97,772 calories. Had Kaiser experimented with purer material this result might have been different. It is one of the elements like sodium, etc., which, when brought into contact with water, decomposes it to form hydrogen gas and metallic oxide.[77 A, 16 (p. 182), 59 (p. 164)] Upon warming powdered titanium, it unites readily with oxygen with brilliant scintillation, and the reaction is so energetic that it propagates itself thereafter, even if removed from contact with the flame.[16—p. 181] Woehler states that he does not know of another body which burns as energetically in oxygen as titanium, and furthermore, that metallic titanium mixed with oxide of copper and warmed, reduces the copper to the metallic state with the evolution of great heat.[16—p. 182] It forms many oxides, such as TiO, Ti_2O_3, TiO_2, etc., of which TiO_2 is the most common.[1 (p. 122), 79, et al] It also forms oxy-chlorides.[70] The oxide of titanium is reduced by aluminum with evolution of heat.

Titanium has a greater affinity for nitrogen than has any other known element. When heated to 800° C. (1,472°F.) in an atmosphere of nitrogen gas, it burns brilliantly with incandescence. Roberts-Austen[73] states that this is the only known instance of vivid combustion in nitrogen. It forms several nitrides, such as Ti_3N, TiN, Ti_2N_2, Ti_3N_4, Ti_3N_5, etc.[16 (p. 180), 70, 116] The massive and fused titanium is not so active in uniting with nitrogen as the powder.[18] Titanium oxide, when heated in the presence of nitrogen, decomposes into titanium nitride;[79 (p. 480), 79 A, 18, 116, 4]—and titanium cyanide[1 (p. 131), 16, 71, 79 B] if carbon also is present. It is in this way that the various mixtures of nitride and cyanide, somewhat vaguely termed "cyano-nitride of titanium," are produced in the blast-furnace and elsewhere. Even titanium cyanide will absorb nitrogen with the evolution of heat.

Titanium also forms a stable carbide, TiC, which has been found in pig-iron and ferro-manganese, but which is said to be almost absent from ferro-titanium, that is to say, from pig-iron when the titanium is large in amount—say more than 10 per cent.[81A] Indeed, the action of titanium is to precipitate carbon from cast-iron and steel.[81A]

Titanium forms one stable sulphide :—TiS_2.[1—p. 182] Blair and Shimer found crystals of the following composition in a gray pig-iron :[81B] Ti, 62.82; S, 22.64; C, 9.82; Fe, 1.82; undetermined, 2.90 per cent. I am inclined to think that titanium sulphide is not readily soluble in molten iron and steel, but, like manganese sulphide, it separates from the molten bath. But the action seems to be more rapid in the case of titanium sulphide than in the case of manganese sulphide, because Goldschmidt[122] and Treuheit[126] found the amount of sulphur decreased after a brief titanium treatment.

Titanium unites with chlorine, fluorine, iodine, hydrogen, etc.; also it forms a silicide, aluminide, boride, etc., but has only slight affinity for phosphorus and arsenic.[1 et al.]

The oxide of titanium acts as an acid flux and unites with iron oxide, aluminum, calcium and other metals to form titanates.[81C] These titanates apparently lower the melting-point of silicates of the same metals; *i. e.*, the dirt and slags occluded in iron and steel.[81C]

TITANIUM IN PIG-IRON.

Some writers have placed a good deal of emphasis on the circumstance that certain grades of pig-iron which have borne an unusually good reputation for strength and toughness for a long period of years, have been thought to contain traces of titanium, or else have been smelted from mixtures containing titaniferous ores. Rossi[3—pp. 102-3] gives a list of references relating to this subject, and a number of text-books freely quote such statements.

It is not to be denied that a reputation for excellence extending over a long period of years forms some basis for such an opinion, as, for example, the titaniferous charcoal pig-iron of Turrach, which has borne an. excellent reputation for 250 years.[64] Neither is it to be denied that some of the most careful metallurgists of the nineteenth century have attributed the excellence of certain grades of pig-iron to titanium, or have, at least, attributed excellence to titaniferous pig-iron; for example: Bell,[52—pp. 750-1] Rivot,[1—pp. 156-7] Osborn,[56—p. 475] Crookes and Roehrig,[55] Akerman,[57] Bauerman,[59—p. 53] Raymond,[63 A] and others. Finally, it is admitted that the presence of titanium in the blast-furnace, or the presence of compounds of titanium in pig-iron, might have the effect of opposing the entrance of nitrogen to the metal, which would, doubtless, increase its strength and decrease its brittleness. Nevertheless, all this evidence is necessarily circumstantial, and in case of a conflict of testimony, should be inclined to place more weight upon comparative tests of cast-iron, such as those given in a later section of this paper.

Titanium in Steel.

What is now often called "titanium-steel," is not a true alloy-steel in the same sense as nickel-steel, manganese-steel, etc., because the titanium is added for the purpose of cleansing and purifying the metal, and not with the expectation of producing new properties by its direct influence. On the contrary, when sufficient titanium has been added to produce a direct influence, the result has not always seemed favorable.[86] Indeed, we may make excellent titanium-steel in which not a trace of titanium can be found—it already having performed its scavenging work and passed out of the bath together with the impurities it was commissioned to remove. The case is somewhat similar to that of manganese, which is added to almost all steels for the purpose of removing oxygen and sulphur, and neutralizing the effects of any traces of those elements which remain thereafter. Small doses of manganese are used for this purpose. However, when we wish to take advantage of the direct effect of manganese, we add about 15 per cent. of the metal, and then get an entirely new result, which makes manganese-alloy steel. Most of the titanium-steel belongs in the first class, and not in the true alloy-steel class. Indications are not wanting, however, that the direct effect of titanium on steel and iron may have its advantages,[100-104 B] and the future will perhaps see titanium alloy-steel used for many purposes.

Even when used for cleansing purposes, titanium hardens, strengthens, and toughens steel, and this seems to apply to crucible-,[91 A] and even electric steel,[91 A] as well as to Bessemer [94,97,99] and open-hearth.[98,103] Touceda tested two steels in a Wright-Souther alternate testing-machine: one of them had been treated with 0.25 per cent. of the usual 10 per cent. grade of ferro-titanium alloy, and the other was like it but without having been so treated. The untreated steel stood 2,676,000 revolutions at a maximum fiber-stress of 38,872 lb.; the treated steel stood 18,274,000 revolutions at a fiber-stress of from 38,872 to 45,939 lb. Waterhouse tested the segregation of steel which had been treated with titanium, and that which had not been so treated, and found a reduction of segregation in the former.[102]

TITANIUM AND NON-FERROUS METALS.

Titanium hardens and strengthens aluminum and increases its toughness and resiliency. It is stated that the addition of titanium, chromium, and copper to aluminum gives some of the hardest and toughest light alloys yet produced. Some years ago, small percentages of titanium were added to aluminum to make an alloy which was used by European bicycle manufacturers; the strength of the aluminum was increased from 23,300 to 73,500 lb. per sq. in.[106] Richards states that titanium has been alloyed with aluminum in proportions up to 7 per cent., but that 2 per cent. gave the best result. This alloy had an elasticity comparable to spring-brass, but it corroded badly, and so has disappeared from the industrial world.[107] Copper is greatly improved by the addition of titanium in small amounts to remove oxygen and gases, and this is a branch of the industry which is now undergoing rapid development.

THE MANUFACTURE OF TITANIUM AND OF FERRO-TITANIUM.

The manufacture of titanium in a substantially pure state by a process cheap enough to be commercially satisfactory, has apparently not yet been accomplished. Moissan's method of reducing the oxide by carbon in an electric furnace seems industrially impracticable; the alumino-thermic process of Goldschmidt incorporates aluminum as an impurity; finally, any smelting process not carried on *in vacuo* gives the molten metal an opportunity to absorb material amounts of nitrogen and oxygen. Numerous wet processes have been developed in laboratories, and have produced samples of titanium of varying degrees of purity from 95 per cent. upward, but these are all too costly to be adopted commercially unless the use to which the metal is to be put will justify a high price. Weiss and

Kaiser[4] and Hunter[116] have described these several methods in some detail, and the latter has obtained a product in which he found by two different methods of analysis nearly 100 per cent. of Ti. His process, in brief, is the reduction of titanium tetra-chloride, $TiCl_4$, by sodium at a red heat. Before his experiments were made, Georges and Staehler[114] had reduced the tetra-chloride by means of hydrogen, but sodium would of course give a purer product, although a more costly one. Indeed, the reduction by means of sodium is not new with Hunter,[2] but it, like all these laboratory methods, is far too costly to produce a metal that can be used in the manufacture of iron and steel. For this purpose we must employ some process comparable in cost with the Goldschmidt thermit process, or the ferro-titanium process of Rossi, which two methods of manufacture are the only ones which have been used on a large scale in connection with metallurgy, so far as I know, although Huppertz[20] and Borchers,[118] Lodyguine,[111] and Gin[112] have each developed processes which seem to have important possibilities.

The Goldschmidt thermit process, as applied to titanium reduction, is not a simple application of the base reaction, as follows:

$$3 : TiO_2 + 2 : Al_2 = 2 : Al_2O_3 + 3 : Ti,$$

because this would not evolve sufficient heat; one employs instead a mixture of iron thermit with titanium thermit in such proportions that, when ignited, a ferro-titanium will be produced containing Fe, 75; Ti, 25 per cent. thus:

$$4 : Fe_2O_3 + 4 : Al_2 = 4 : Al_2O_3 + 4 : Fe_2;$$
$$3 : TiO_2 + 2 : Al_2 = 2 : Al_2O_3 + 3 : Ti.$$

The heat evolved by the first reaction augments that evolved by the second reaction, and this makes an energetic exothermic effect. Ferro-titanium could, if desired, be made in this manner independently and then added to iron or steel, but it is not applied this way industrially. Instead, one step is made of the two operations, and the mixture of iron- and titanium-thermits is added to the molten iron or steel which is to be treated, so that the alloy will be reduced *in situ*.

Rossi makes ferro-titanium containing from 10 to 15 per cent. of titanium, and from 5 to 8 per cent. of carbon, and the residue iron, by heating a pulverized mixture of titaniferous iron-ore and charcoal to a temperature of not less than 1,927° C. (3,500° F.). This operation is performed in an arc electric furnace of the Siemens type. If an alloy low in carbon is desired, a slightly different form of furnace is employed, in which graphite does not come in contact with the molten alloy, and the reduction is performed by liquid aluminum, as follows: Scrap-aluminum is charged and melted, followed by the desired amount of scrap-iron. The iron forms a liquid bath upon which the aluminum floats. Upon this layer of aluminum is charged rich titaniferous ores, or else a titaniferous slag consisting of calcium titanate, the manufacture of which is performed in an auxiliary furnace. The reaction between the melted aluminum and the titanium oxide occurs at once at the surface of contact, and has so much energy of its own that little electricity is required thereafter to keep the temperature up to the reaction-point. The reduced titanium unites with the iron bath. The rich slag used in this operation is also used when desired instead of titaniferous iron-ores in making the high-carbon grade of ferro-titanium. It is made by smelting titaniferous iron-ores with a limited amount of charcoal and at a temperature sufficiently elevated to reduce iron and silicon, but not to reduce titanium. Pig-iron substantially free from titanium is thus produced, and contains the bulk of the impurities in the ores, such as sulphur, phosphorus, etc. The titanium oxide is fluxed with lime to form a slag consisting chiefly of calcium titanate, CaO, TiO_2.

RESEARCHES UPON THE EFFECT OF TITANIUM ON CAST-IRON.

In some of his experiments, Rossi appears to have added 5 per cent. of a pig-iron containing 0.2 per cent. of titanium to a non-titaniferous pig-iron, and increased the transverse strength thereof from 3,390 to 3,778 lb. per sq. in. Then he added 25 per cent. of the same pig-iron, but the strength was increased only to 3,480 lb. I am inclined to believe that he chilled his iron when using the larger amount of titaniferous pig, or else that he raised the freezing-temperature without equally raising the casting-temperature, and thus destroyed in part the good effect on strength which the titanium treatment would have had.

This matter of the casting-temperature of cast-iron is one which a great many investigators of titanium treatment seem to have overlooked; indeed, it is one upon which little scientific thought appears to have been expended, for I have found few published data upon it. However, Longmuir[121] has studied the effect of casting-temperature on tensile strength, and his figures will serve as a guide to show us the importance of this question. I have averaged the record of about 42 tests published by him in 1903 and 1904, with the following result: The strength of iron cast at the correct temperature (estimated by Longmuir as about 1,230° C. = 2,246° F.) averaged 23,985 lb. per sq. in.; that of iron cast too hot (estimated by him as about 1,320° C. = 2,408° F.), was 16,100 lb., equivalent to a loss of 33 per cent.; that of iron cast too cold (estimated as about 1,120° C. = 2,048° F.), was 17,781 lb., or a loss of about 25 per cent. Therefore, if we are aiming to improve strength, we must guard against casting the iron either too hot or too cold, and this matter is equally important whether we operate by means of titanium or in any other way.

Rossi's experiments above noted appear to have been made in crucibles and in the iron casting-ladle. He supplemented them by others in which he used 4 per cent. of a ferro-titanium alloy containing from 10 to 15 per cent. of Ti. By this means he increased the transverse strength of cast-iron by from 20 to 25 per cent., and the tensile strength by from 30 to 50 per cent.

These tests were followed by those of Schiemann,[120] who added 5 per cent. of ferro-titanium thermit to a series of cast-irons, and increased the tensile strength by varying proportions between 30 and 50 per cent.

Rossi made some experiments in 1905, in which he added 1 per cent. of a ferro-titanium alloy containing 12 per cent. of titanium to cast-iron and increased the transverse strength by from 17 to 23 per cent., and the tensile strength by from 6 to 29⅛ per cent.

In the same year, Goldschmidt[122] reports a series of tests in which he added 0.1 per cent. of titanium to iron by means of thermit, and obtained increases in the transverse strength of from 1 to 17 per cent., and in the tensile strength of from 4 to 16 per cent.

In 1908, Prof. Martin Hokanson,[123—pp. 62-3] of the Carnegie

Technical Schools, found the compressive strength of chilled cast-iron which had been treated with ferro-titanium alloy to be 298,000 lb. per sq. in., as compared with 173,000 lb. for the same metal untreated. The hardness of the treated metal was 557 (Brinell), as compared with 455 for the untreated iron.

Dr. Richard Moldenke[123] made a series of transverse tests on white and gray cast-irons in 1908, with the following results:

Gray Irons.	Lb. Per Sq. In.	White Irons.	Lb. Per Sq. In.
Untreated iron (average of 9 tests),	2,020	(Average of 8 tests).......	2,050
Plus 0.05 p.c. of Ti[a] (average of 4 tests),	3,100	(Average of 11 tests)........	2,400
Plus 0.10 p.c. of Ti (average of 3 tests),	3,030	(Average of 9 tests)	2,420
Plus 0.05 p.c. of Ti (average of 6 tests),	3,070	(Average of 10 tests).......	2,400
Plus 0.10 p.c. of Ti (average of 6 tests),	2,990	(Average of 10 tests).......	2,520
Plus 0.15 p.c. of Ti (average of 4 tests),	3,190		

[a] Titanium added in form of ferro-titanium alloy.

Average of treated gray cast-iron, 3,070, equivalent to increase of 52 per cent.

Average of treated white cast-iron, 2,430, equivalent to increase of 18 per cent.

Moldenke added ferro-titanium alloy equivalent to 0.10 per cent. of titanium to the bed of a cupola, but the results of tests of the metal therefrom were so irregular that he judged an average to be unrepresentative. I have duplicated this result several times, and am inclined to believe that the oxidizing influence brought to bear on the bed of a cupola makes the use of titanium there of uncertain value as far as strength is concerned. The whole question of the use of titanium in the cupola requires investigation. The ferro-titanium alloy melts at a higher temperature than the iron itself, and therefore gets nearer the tuyeres before it begins to trickle down into the hearth. If, during this transit, it meets with slag, its value will be lost as far as the iron is concerned, because it will attack the oxygen in the slag and unite with it, thus being used up. Cupolas must be constantly drained clear of slag when titanium-alloy is melted with cast-iron therein. We must go even further than this precaution to prevent a layer of slag accumulating on top of the metal and intercepting the drops of melted alloy, because constantly draining the cupola will not keep the slag away provided the level of the iron bath rises and falls several inches at intervals, as, for example, when the iron is collected inside the cupola instead of in the ladle.

Feise[124] gives the following table of tests made on gray cast-irons:

	Transverse Strength.	Deflection.
Without titanium	1,230 kg/qcm.	0.046 mkg.
Plus 0.25 per cent. of Ti	1,660 kg/qcm.[a]	0.056 mkg.
Plus 0.50 per cent. of Ti	1,410 kg/qcm.	0.053 mkg.
Plus 1.00 per cent. of Ti	1,340 kg/qcm.	0.066 mkg.

[a] Equivalent to 35 per cent. increase in strength.

These results corroborate the observations of others: that the maximum effect on strength appears to be obtained with about 0.10 per cent. of titanium.

The most unsatisfactory results I have found were published by Treuheit,[126] and report an elaborate series of tests on cast-iron treated with ferro-titanium thermit and ferro-titanium alloy, respectively. The temperatures were measured by a pyrometer, and the resulting metal was tested by both tensile and transverse methods. In some cases the treated metal was weaker than that which had not been treated, and sometimes a little stronger, but never enough different to indicate anything especially interesting. In examining the record of this work, one does not have to look far to find a reason for the unsatisfactory result. The temperatures at which the iron was cast were widely variant, sometimes being above 1,300° C. (2,372° F.), which is shown by the work of Longmuir to be detrimental to quality. One might also infer from the author's words that he did not allow the metal to stand long enough after adding the titanium, so that the latter could exert its cleansing influence; at best, his record is not clear on this point. Three minutes should be allowed the titanium to act upon cast-iron before it is teemed, and more if the alloy is not then absorbed, as indicated by the pasty state of what remains. If the iron has not heat enough to permit this 3-min. wait, then it should have been produced in hotter condition in the first instance; but, in any event, if it cannot be held 3 min., and still poured at the correct teeming-temperature, one must not blame the lack of success on the treatment, but on the manner in which the treatment is performed. I have failed in test after test because the metal had to be carried some distance from the cupola before it could be treated, or because it was at about the correct pouring-temperature before the 3-min. wait, and, therefore, too cold thereafter. If the titanium treatment is advantageous to cast-iron, it must also be advantageous to perform the treatment properly.

When titanium unites with the oxygen or nitrogen in iron or steel, there is, unquestionably, an evolution of heat, which opposes the cooling of the bath during the wait. Fitzgerald[123] showed that the temperature of a bath of cast-iron increased 25° F. during its treatment with 1 per cent. of titanium in the form of ferro-titanium, but that this lasted only 1 min., and then the temperature slowly fell again. The original temperature of the metal must be hot enough to endure this drop and still be at the correct point.

Another mistake made by Trueheit, when treating this iron with ferro-titanium alloy, was in heating the alloy red-hot in the ladle before tapping the iron therein. This heating has the effect of oxidizing the alloy and reducing, or, maybe, nullifying, its cleansing effect; the practice was once recommended by the manufacturers of the alloy, but is now abandoned. Furthermore, the alloy should not be placed in the bottom of a ladle, because its infusibility causes it to become pasty and stick to the lining without dissolving in the bath. It should be thrown into the ladle after there is a layer of iron upon which it can float, and then the in-pouring stream of metal will stir it well and assist its absorbtion.

If Trueheit's experiments were the only ones to guide us, or if they were confirmed by other results skillfully obtained, then we might doubt the value of titanium to cast-iron, but the many favorable researches quoted herewith would make us question Trueheit's conclusions, even if we could not point out very definitely the cause of his ill-success.

C. H. Gale,[127] one of the most experienced American malleable cast-iron manufacturers, reported, in 1911, the results of many tests made on the treatment of malleable cast-iron with ferro-titanium alloy. In his first test, Gale added, first 1.25 and then 2.50 per cent. of the alloy in the ladle, with the consequence that the iron was teemed much too cold, and, also, it proved to be mottled, instead of having the usual, and desired, white fracture. Both of these conditions would, of course, decrease the strength of the metal; nevertheless, the treated bars in one set averaged slightly stronger than those untreated.

In another set they were about 19 per cent. weaker in tensile test. The transverse tests of the treated metal proved it to be from 5 to 30 per cent. stronger than the untreated metal. Nevertheless, the result indicated that such a ladle treatment of malleable cast-iron did not promise much success unless the silicon were reduced so as to prevent mottled-iron being produced, since titanium causes a precipitation of graphite, and unless the metal were increased in temperature so as to wait during the time of treatment and still be at a good heat for pouring. (I may say here that a test which I made on malleable cast-iron fully corroborates this conclusion.)

Gale then made a series of tests, adding the alloy in the furnace from 30 to 45 min. before tapping, but it scarcely seems likely that this would have as marked a result as if the alloy had been put in the furnace as soon before tapping as practicable, because titanium oxidizes so quickly that its effect would be lost in a comparatively short time. However, the transverse tests show an improvement on the average of 15 per cent. in the bars treated with 0.30 per cent. of the alloy, and 17 per cent. in the bars treated with 0.60 per cent. The tensile tests were not so favorable—12 per cent. and ⅛ per cent. respectively—but Gale cautions us against taking the tensile tests too seriously, because the bars were round in section and may not have been sound. In the discussion of those results, W. D. Alexander gave the report of a long series of tests made by him upon the treatment of malleable cast-iron with the usual ferro-titanium alloy—*i. e.*, that containing from 10 to 15 per cent. of titanium—in proportions from 0.47 to 1.75 per cent. He added the alloy in the hand-ladle caught from the furnace, and thus there was a great cooling of the metal, especially when using the larger amounts of alloy. The ductility of the metal was then tested, after annealing, by upsetting bars with a 10-lb. sledge. The metal treated with 0.47 per cent. of alloy showed an increase in ductility by this test of 105.3 per cent.; that treated with 0.75 per cent. of alloy improved 65.7 per cent.; that with 1 per cent. of alloy decreased 10.5 per cent.; that with 1.25 per cent. of alloy decreased 69.1 per cent.; that with 1.50 per cent. of alloy decreased 59.4 per cent.; and that with 17.5 per cent. of alloy decreased 128.2 per cent. Evidently the best result would be obtained with 0.47 per cent. of alloy or less, unless some means could be had of heating the metal hotter before the treatment.

In 1911, Geiger[128] published his book on iron- and steel-foundry practice, in which he devotes two sections to the effect of titanium on the ferrous metals. Finally, he reports a series of tests on iron with titanium in proportions of 0.04, 0.07, and 0.10 per cent., equivalent to 0.40, 0.70, and 1.00 per cent. of the usual ferro-titanium alloy, respectively. The respective improvements in tensile strength were 12, 24, and 26 per cent.; of transverse strength, 6, 9, and 10 per cent.; and of deflection in transverse test, 9, 11, and 6 per cent.

Thomas D. West,[129] by adding 12 oz. of the usual ferro-titanium alloy to a ladle of cast-iron containing 225 lb., obtained an increase of 27 per cent. in the transverse strength, as an average of five tests; by adding 22 oz. of ferro-titanium alloy to 225 lb. of iron, he obtained an increase in transverse strength of 32 per cent., as an average of five tests.

Osann [130—p. 118] reports on the improvement in the compressive strength of cast-iron for car-wheels; in one case an increase of 18 per cent. was obtained by treatment with titanium, and, in another case, of 75 per cent.

In my own tests, which included both white and gray cast-iron, the iron was melted in a cupola (except for one test on malleable cast-iron, referred to just above), and cast into round bars 18 in. long and from 1 to 1.3 in. in diameter. They were tested transversely on supports 12 in. apart, according to the standard testing-specifications of the American Society for Testing Materials and the American Foundrymen's Association. The figures reported below indicate moduli of rupture in pounds per square inch. During the research, I made several errors in operation, due to ignorance of the conditions which should have prevailed, but the reports of these improperly-performed tests are here given with the others, because, if they teach nothing useful, they at least emphasize the need of performing the titanium treatment correctly. The treatment was made with the usual ferro-titanium alloy containing from 10 to 15 per cent. of titanium, and the additions are recorded in percentages of alloy added, instead of percentages of titanium.

Considering first some of the incorrectly-performed tests, I made two tests charging the alloy with the iron of the bed of the cupola, hoping that the deoxidizing effect of the titanium would overcome the oxidizing effect of the blast. This hope was not realized, as will be seen below:

Test No. 1 (*Cupola Bed*).

Iron without titanium (average of 4 tests)................ 42,175 lb.
Iron with 0.50 p. c. of alloy on bed (average of 4 tests), 40,493 lb., decrease of 4 p.c.

This failure of the titanium-treated iron to exceed in strength that not treated is the more significant because the same iron showed an increase of 6 per cent. in strength when 0.5 per cent. of alloy was added to the ladle, although it had to be carried some distance in the foundry before pouring, and was therefore too cold after the treatment. The untreated iron was not bed-iron, and this difference is important.

Test No. 2 (*Cupola Bed*).

Iron without titanium (average of 4 tests)...............41,520 lb.
Iron with 0.40 p. c. of alloy on bed (average of 4 tests), 40,610 lb., decrease of 2 p.c.

In spite of the apparent decrease in the treated metal, the titanium is shown to have been beneficial, because, normally, bed-iron would have been from 20 to 30 per cent. weaker than the other. This same iron gave an increase in strength of 16 per cent. when treated with 0.5 per cent. of alloy in the ladle, and of 9 per cent. when treated with 2 per cent. of alloy in the ladle.

I tried the experiment of adding the alloy in a cupola from which the slag was not drained continuously. The results were irregular and inconclusive, as shown below:

Tests Nos. 3, 4, 5, *and* 6 (*Wasted in Cupola Slag*).

Iron without titanium (average, 4 tests).........42,378 lb.
Iron with 0.5 p. c. of alloy (average, 4 tests)....35,775 lb., decrease of 15 p.c.

Iron without titanium (average, 6 tests)...........48,517 lb.
Iron with 0.5 p. c. of alloy (average, 6 tests.......54,830 lb., increase of 13 p.c.

Iron without titanium (average, 8 tests)............44,893 lb.
Iron with 0.5 p. c. of alloy (average, 8 tests)......42,856 lb., decrease of 5 p.c.

Iron without titanium (average, 2 tests)............55,280 lb.
Iron with 0.5 p. c. of alloy (average, 2 tests).....52,570 lb., decrease of 5 p.c.

Several tests were made on iron which had to be carried some distance while molten, and which before treatment was at a good temperature for teeming, but cooled too far during the 3-min. wait so that it was too cold after treatment. At first the treatment was performed in a ladle holding about 225 lb. in which the loss of heat by radiation was very large. The results were:

Test No. 7 (*Iron Too Cold*).

Iron containing: Si, 2; S, 0.075; P, 0.73; Mn, 0.57 per cent.
Iron without titanium (average 2 tests)......................55,280 lb.
Iron with 0.5 p. c. of alloy in ladle (average 2 tests)...50,100, decrease 10 p.c.

Test No. 8 (*Iron Too Cold*).

Iron containing: Si, 1.25; S, 0.040; P, 0.20; Mn, 0.70 per cent.
Iron without titanium (average, 3 tests)..................49,730 lb.
Iron with 0.25 p.c. of alloy in ladle (average, 4 s),test 44,680 lb., decrease 10 p.c.
Iron with 0.5 p. c. of alloy in ladle (average, 2 tests), 46,000 lb., decrease 8 p.c.
Iron with 1 p. c. of alloy in ladle (1 test only).......54,300 lb., increase 9 p.c.
Iron with 2 p. c. of alloy in ladle (1 test only).......47,850 lb., decrease 4 p.c.

If the metal had been poured as cold as this, and had not been treated, the decrease in strength would have been fully 25 or 30 per cent. (see Longmuir, *ante*).

After this the iron was treated in a 1-ton ladle, in order to reduce radiation of heat, but still the metal was cool after the 3-min. wait, and the results were only partly satisfactory.

Test No. 9 (*Iron Cool*).

Iron containing Si, 1.50; P, 0.75 per cent.
Iron without titanium (average, 6 tests)..................48,517 lb.
Iron with 0.5 p.c. of alloy in ladle (average, 5 tests) 49,280 lb., increase 2 p.c.

Test No. 10 (*Iron Cool*).

Iron containing Si, 1.30; P, 0.37 per cent.
Iron without titanium (average, 8 tests)..................44,893 lb.
Iron with 0.5 p. c. of alloy in ladle (average, 7 tests), 48,419 lb., increase 8 p.c.

Test No. 11 (*Iron Cool*).

Iron containing Si, 0.90; P, 0.20 per cent.
Iron without titanium (average, 5 tests)...............42,175 lb.
Iron with 0.5 of alloy in ladle (average, 3 tests).....44,525 lb., increase 6 p.c.

Using iron which could be treated and poured out without intervening delays—except, of course, the 3-min. wait for treatment—the following results were shown:

Test No. 12.

Close-grained gray-iron, P more than 1 per cent.

Iron without titanium (average, 4 tests)................35,810 lb.
Iron with 0.20 p. c. of alloy in ladle (average, 4 tests), 39,640 lb., increase 11 p.c.
Iron with 0.64 p. c. of alloy in ladle (average, 4 tests), 40,545 lb., increase 13 p.c.

Had I thought in time to make such arrangements that both treated and untreated iron should be poured at exactly the same temperature, the test would have been still more representative. As it was, the treated iron was poured a little cooler than the other.

Test No. 13.

Gray-Iron.

Iron without titanium (average, 4 tests)................41,520 lb.
Iron with 0.5 p. c. of alloy in ladle (average, 4 tests), 48,350 lb., increase 16 p.c.
Iron with 2 p. c. of alloy in ladle (average, 7 tests), 45,390 lb., increase 9 p.c.

Test No. 14.

White-Iron.

Iron without titanium (average, 2 tests)................32,800 lb.
Iron with 0.5 p.c. of alloy in ladle (average, 2 tests), 37,450 lb., increase 14 p.c.

Two tests were made by adding the alloy in the cupola, the slag being continuously drained:

Test No. 15.

White-Iron.

Iron without titanium (average, 2 tests)................32,800 lb.
Iron with 0.3 p.c. of alloy in cupola (average, 8 tests), 47,040 lb., increase 43 p.c.
Iron with 0.45 p.c. of alloy in cupola (1 test only)....37,600 lb., increase 12 p.c.
Iron with 0.60 p.c. of alloy in cupola (1 test only),[a] 32,950 lb., increase 0.5 p.c.

[a] Last charge in cupola; iron badly oxidized.

Three tests were then made on iron containing between 3 and 4 per cent. of phosphorus, with the following results. The iron was liquid when poured, but not very hot.

Tests Nos. 16, 17, *and* 18.

High-Phosphorus Iron.

Iron without titanium (average, 3 tests)................46,583 lb.
Iron with 0.37 p c. of alloy in ladle (average, 2 tests)..49,900 lb., increase 7 p.c.

Iron without titanium (average, 2 tests)...............47,600 lb.
Iron with 0.5 p. c. of alloy in ladle (1 test only)......54,650 lb., increase 14 p.c.

Iron without titanium.......................................42,450 lb.
Iron with 1 p.c. of alloy in ladle (average, 2 tests)...46,385 lb., increase 10 p.c.

Iron containing P about 9 per cent. was tested:

Test No. 19 (*Very High-Phosphorus Iron*).

Iron without titanium (average, 2 tests)..................41,900 lb.
Iron with 0.5 p.c. of alloy in ladle (average, 2 tests), 42,630 lb., increase 2 p.c.

The significance of these results is that iron may be increased in strength up to 43 per cent. by the correct use of titanium. However, to accomplish the best results, the titanium must not only be in correct proportions, but the iron must be given a fair chance by allowing the titanium a full opportunity to become thoroughly incorporated in it (not less than three minutes treatment), so that it is neither wasted in the slag nor left unmelted in the ladle, and by insuring that the iron is poured neither too hot nor too cold.

SUMMARY OF RESULTS.

A summary of the results obtained by different investigators follows:

		Increased.	
		Tensile Strength Per Cent.	Transverse Strength. Per Cent.
Rossi,	Alloy	30 to 50	20 to 25
Schiemann,	Thermit	30 to 50	
Rossi,	Alloy	6 to 30	17 to 23
Goldschmidt,	Thermit	4 to 16	1 to 17
Moldenke,	Alloy	......	18 (white)
Moldenke,	Alloy	......	52 (gray)
Guillier,[91B]		......	49 (malleable cast-iron)
Guillier,[91B]		......	14 to 30 (gray)
Feise,	Alloy	......	35
Treuheit,	Thermit and alloy	No important change.	No important change.
Gale,	Alloy	⅛ to 12	15 to 17[a]
Alexander,	Alloy	(Increased ductility.[a])	
Geiger,	Alloy	13 to 26	6 to 10
West,	Alloy	......	27 to 32
Stoughton,	Alloy	......	10 to 43[a]

[a] When treatment was properly performed; otherwise, results different and variable.

These results indicate that it is entirely possible, by correct treatment with ferro-titanium alloy, to increase the strength of cast-iron by from 30 to 50 per cent. If this can be done part of the time, then it only remains to determine the correct conditions in order to attain equal success all the time. Some of these conditions I have pointed out in the foregoing pages, but I do not consider that the problem is completely solved as yet. What has been learned is published now in the hope that it will inure to the benefit of makers and users of cast-iron.

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[TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS.]

Surveying and Sampling Diamond-Drill Holes.

BY E. E. WHITE, ISHPEMING, MICH.

(Cleveland Meeting, October, 1912.)

In August, 1911, I read a paper before the Lake Superior Mining Institute[1] on surveying and sampling diamond-drill holes. The present paper gives a more thorough description of these methods, together with a few changes and notes suggested by further experience. Since August, 1911, the methods, used constantly by the Cleveland-Cliffs Iron Co., have been developed to a higher degree of efficiency, and have given great satisfaction.

Until a few years ago but little attention was paid to the determination of the actual course of drill-holes, to the accurate sampling and analysis of the material, or to the scientific location of the drill-holes. It has now been common practice for several years in many districts to test the inclination of drill-holes by etching glass tubes with hydrofluoric acid, but the direction of deviation from the vertical has not often been determined. Methods of determining the direction by means of a compass were first developed in connection with drilling in Victoria. The latest method used on the Rand in South Africa has been recently described by John I. Hoffman.[2]

In the same paper Mr. Hoffman describes an ingenious method of wedging-off a drill-hole so as to start new holes at several successive points. By this means a hole can be deflected in any desired direction, and the curvature can be controlled to a considerable extent, provided the rock is soft enough to permit the hole to be wedged off. Unfortunately, in the Lake Superior district, the jasper iron-formation is so hard that it is very difficult, if not impossible, to do this. I tried to deflect one hole so as to avoid drill-rods stuck in the hole, by using a

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[1] *Proceedings of the Lake Superior Mining Institute*, vol. xvi., pp. 100 to 120 (1911).

[2] Recent Practice in Diamond-Drilling and Borehole Surveying, *Bulletin No.* 91, *Institution of Mining and Metallurgy* (Apr. 11, 1912).

12-ft. wedge of the hardest steel obtainable, but the bit cut the wedge and hardly touched the jasper. If there had been a dike of softer rock higher up in the hole I might have been able to deflect it at that point. Two years ago I described[3] how the curvature of a diamond-drill hole can be controlled by other means, and gave an example where it had been done successfully. Although the hole in this instance kept to the desired inclination, it failed to intersect the ore encountered in a previous hole because the direction of deviation from the vertical of the previous hole had not been determined, although the amount of deviation had been determined by the usual hydrofluoric acid tests.

Since 1910, I have experimented with two methods of determining the direction of deviation, one applicable to non-magnetic and one to magnetic rock-formations, with very good results. We now test all drill-holes except very shallow ones for both the amount and the direction of deviation, usually at 200-ft. intervals, and we have the satisfaction of knowing very closely the actual position of any ore or other formation encountered. In non-magnetic rock-formations the old method of a compass suspended in gelatine is successfully used, with improvements worked out by George Maas and me, and patented by Mr. Maas (Patent No. 1,003,624, Sept. 19, 1911). In magnetic formations a method of marking the drill-rods is used in connection with hydrofluoric acid tests. This method was, to the best of my knowledge, first applied by John Deacon, Superintendent of the Republic Iron & Steel Co.'s properties at Negaunee, Mich., who used it several years ago in testing diamond-drill holes at the Cambria mine. I found that with great care good results can be obtained by this method, but it is more difficult and expensive than the compass method. It is the only one I know of that is practicable in magnetic formations where the direction of strike underground is unknown. If the strike were known it would be easy to take an impression of the bottom of the hole with a piece of lead and determine the direction of deviation by comparing the direction shown by an acid etching with the strike shown by the piece of core broken from the bottom just before making the test, when made to coincide with the impression in the lead. Jean Florin[4] describes a similar method, using a compass and camera instead of the acid.

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[3] *Engineering and Mining Journal*, vol. xc., No. 12, pp. 546, 547 (Sept. 17, 1910).

Surveying in Non-Magnetic Formations.

Figs. 1 and 2 show the cases used to test for inclination and course, using the latter when more than one test is to be made at one time, since it may be inserted at any point in the drill-rods at the same time that the first case is used at the end of the rods. As it is desired to use as large a glass tube in the case as possible, and as the outside diameter is limited by the size of an E hole, a material was selected which combined the greatest possible toughness and tensile strength with non-mag-

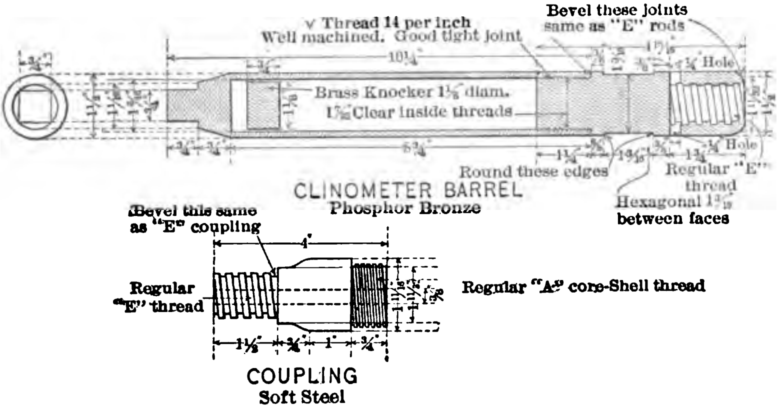

Fig. 1.

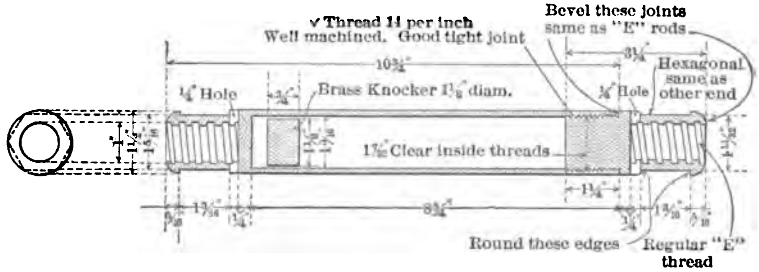

Fig. 2.

Figs. 1 and 2.—Phosphor-Bronze Cases to Test Inclination and Course of Drill-Holes.

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[4] *Annales des Mines de Belgique*, summarized in *Engineering and Mining Journal*, vol. lxxxvii., No. 17, p. 854 (Apr. 24, 1909).

netic properties. Phosphor-bronze was chosen, which is entirely non-magnetic, and which can be obtained with a tensile strength of 60,000 lb. and an elastic limit of 54,000 lb. per sq. in. By using a case of the dimensions given in Fig. 1, a glass tube 1⅛ in. in outside diameter can be used, and according to Nystrom's formula for the collapsing-strength of small tubes, $P = \frac{4Tt^2}{fd\sqrt{L}}$, using a factor of safety of 4, this case should be safe in a hole 2,800 ft. below water-level. A little wicking is used to make a perfectly tight joint.

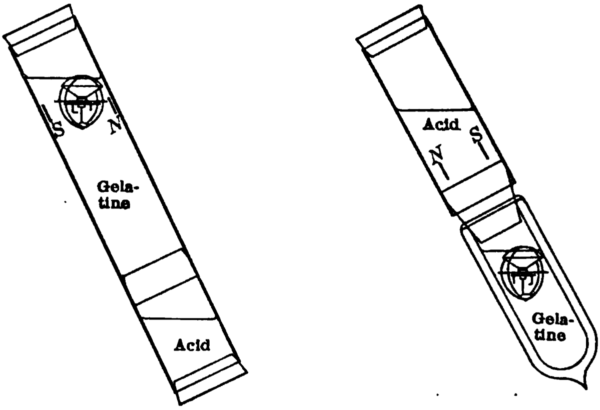

FIG. 3.—MAAS'S PATENT DRILL-HOLE COMPASS IN TUBES, READY TO MAKE TEST.

The compass invented by Mr. Maas, shown in Fig. 3, has the advantage over the old forms of compass used for this purpose that it is pivoted in a cage which prevents it from coming in contact with the glass tube, and insures a free swing in the gelatine. The cage is below and rigidly attached to a cork float.

The most accurate and satisfactory method of testing the course of a hole is to use the compass in a glass tube about 6 in. long, open at each end. A section of rubber stopple is forced into the tube, leaving about 1.5 in. space for acid at one end, and 4 in. space for gelatine at the other. The gelatine may be prepared beforehand, but usually a small weighed portion of dry gelatine is carried to the drill and dissolved on the ground in a given quantity of water, care being taken that the water has no chance to evaporate while dissolving the gelatine. The proportions are so chosen that when dissolved the solution will keep liquid as long as possible after being lowered in the drill-hole, and yet will become perfectly solid when cold. For instance, with Nelson's Improved Brilliant Gelatine we use ⅝ g. and dissolve it in 50 cc. of water. In a hole where the rods can be lowered in 20 min. or less, a 1⅛-in. tube is used with paper wrapping. When it takes from 20 to 30 min. to lower the rods, a 1-in. tube is used with several wrappings of paper. If deeper than this, a thermos bottle is used, and by wrapping with paper the gelatine may thus be kept liquid 50 min. The time the gelatine remains liquid was determined by tests in water cooled by ice to 43° F., the approximate temperature of the underground water in the Lake Superior district.

In the first two cases, when the thermos bottle is not necessary, the dissolved gelatine is poured into the tube, which is then heated by immersing in water heated to boiling by live steam. When hot, the compass is dropped in and a stopple is placed in that end; then about 1 in. of dilute hydrofluoric acid is poured into the other end and that end closed. The tube is then wrapped in paper and placed with gelatine end up in the bronze case, which is attached to the bottom of 20 ft. of brass E rods and lowered into the hole, losing as little time as possible. The brass rods are screwed to the bottom of the regular drill-rods, using an A to E reducing-coupling if the hole is being drilled with A rods. The bronze case and brass rods are made for an E hole, so that they can be used in either case. If two tests are to be made at the same time, another tube and compass are placed in the case shown in Fig. 2 and inserted in the drill-rods at the proper point, using 20 ft. of brass rods on each side.

The tube is left stationary in the hole 50 min. after the rods are lowered, giving the gelatine time to cool and set, and the acid time to etch a good line. Acid diluted with 12 parts of water gives the best results. This solution is prepared in the office and carried to the drill in hard-rubber bottles, which are much more convenient than the paraffin bottles in which the acid is usually furnished.

When the tube is brought to the surface, the positions of the north and south points of the needle are marked on the glass with a diamond point and the tube is washed out. This tube then forms a permanent record of the inclination and course of the hole at the depth at which the test was made.

The thermos bottle, 1⅛ in. in outside diameter, consists of two clear glass walls, with a vacuum between, shown in Fig. 4. When it is necessary to use this, both the hot gelatine and the compass are placed in the bottle, which is then closed by a rubber stopple. The stopple also closes one end of a 1⅛-in. tube 3 in. long, serving to connect the bottle and tube and preserve them in the same relative position shown in Fig. 4. Dilute acid is placed in the tube, the other end closed, and the tube and bottle placed in the bronze case and lowered into the drill-hole. It only takes the gelatine 90 min. to solidify in the thermos bottle, so that if it takes 40 min. or more to lower the rods, the test is usually left in the hole 50 min. after the rods reach the bottom, just long enough to get a good etching. It may be left in the hole over night, but in that case the acid should be more dilute. When the tube is brought to the surface, the north and south points are marked on it, corresponding to the position of the compass-needle in the thermos bottle. The 3-in. tube then forms a permanent record of course and inclination, just as the 6-in. tube does.

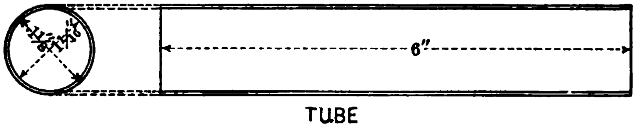

FIG. 4.—THERMOS BOTTLE.

In either case the inclination is read in a goniometer and is corrected for capillarity according to a curve which is prepared for each size of tube by testing tubes at known angles. Fig. 5 shows a curve for 1⅛-in. tubes. It will be noted that for these tubes the correction is only 3.75° at 45°, which is the maximum. The angle can be read to 0.5°, and I feel certain that the results of tests for inclination can be relied upon to within 1°.

To determine the course of the hole, the tube is placed in a special two-circle goniometer, Fig. 6, with both circles set at 0°, so that the tube is vertical, and the vertical circle parallel to the crosspiece over the top of the instrument. If the inclination of the hole is steep, the tube is twisted until the etching shows the dip to be either directly towards or away from the eye; that is, until the cross-thread bisects the ellipse etched on the glass. If the inclination is shallow, it is more accurate to twist the tube so that the dip is to the right or left of the observer and in the plane of the vertical circle. The vertical circle, holding the glass tube, is then turned until the north and south marks on the tube are in line with the cross-piece, when a pointer on the horizontal circle reads the course of the drill-hole. Fig. 3 shows tubes with acid, gelatine, compass, and north and south points marked, just as they are taken from the drill-hole.

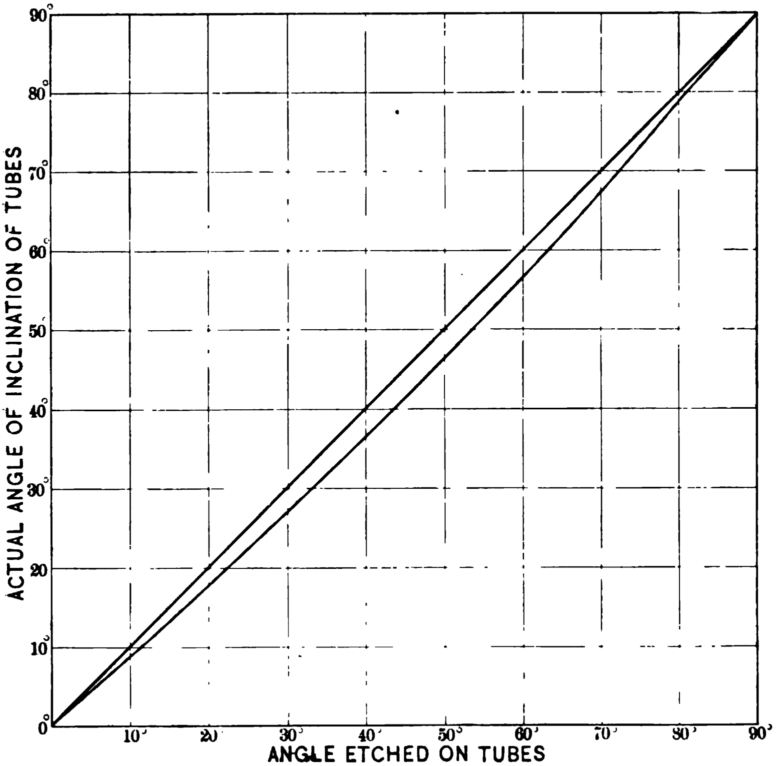

Fig. 5.—Curve of Correction for Capillarity in Testing Inclination of Drill-Holes.

We have found the method described above very successful, and two tests at the same point almost always agree to within a few degrees. When this is not the case, more tests are made, and so far we have always been able to ascertain which are wrong. We have made tests at a depth of 2,000 ft., but it would probably be difficult to go much deeper without using more insulating wrapping around the thermos bottle than is possible with the bottle and case now used.

The precautions to be taken are: 1, that the compass swings perfectly freely, and does not catch on the cage; 2, that the gelatine keeps liquid long enough; 3, that the compass is not used when there is much local magnetic attraction in the rock-formation; and 4, that the compass is not affected by the steel drill-rods or casing, or by other iron in the hole. The first and second precautions are easily taken; the third can be judged only by a knowledge of the formation and by taking tests at different depths, which, if concordant, would indicate that there is no appreciable magnetic attraction. The fourth pre-

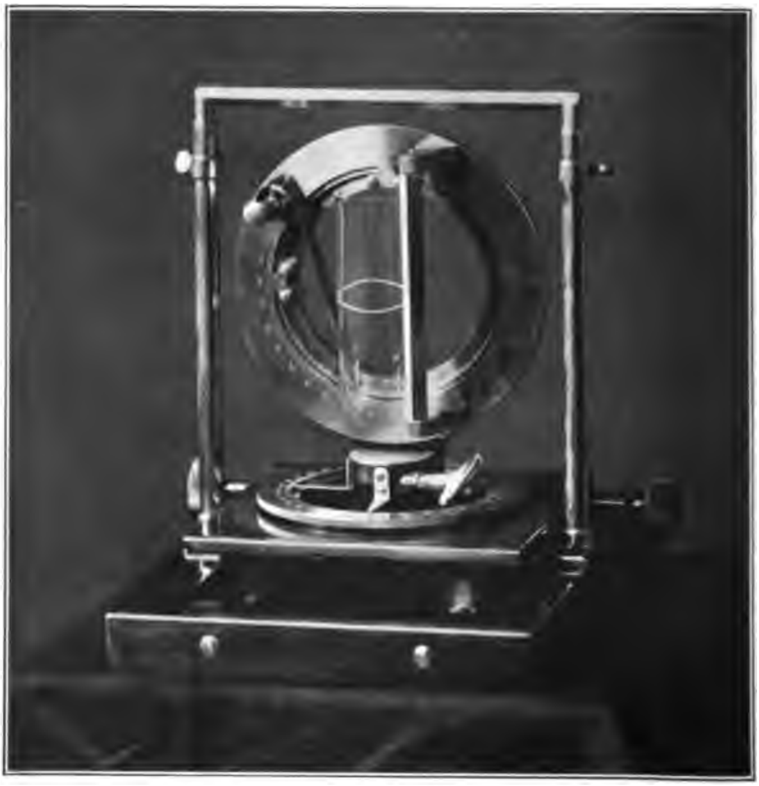

Fig. 6.—Two-Circle Goniometer to Measure Inclination and Direction of Drill-Holes.

caution is important. We use two 10-ft. lengths of brass rods, and so have no iron within 20 ft. of the compass. Tests with 10, 20, 30, and 40 ft. of brass rods at the same depth gave the same reading in a hole dipping 50° N. 45° E., so that a length of 20 ft. is conservative. The results of a second test with shorter lengths of brass rods at another point gave:

Length of Brass Rods. Feet.	Inclination. Degrees.	Apparent Course. Degrees.	Length of Brass Rods. Feet.	Inclination. Degrees.	Apparent Course. Degrees.
1	62	N. 6 E.	7	62	N. 41 E.
2	......	N. 21 E.	8	......	N. 23 E.
3	.....	N. 39 E.	10	......	N. 48 E.
4	......	N. 35 E.	12	......	N. 50 E.
5	......	N. 42 E.	20	......	N. 47 E.
6	......	N. 42 E.			

These data show that the true course is between N. 47° E. and N. 50° E., and that 20 ft. of brass rods is conservative. It is impossible to obtain results accurate to closer than 2° or 3°, because of the small size of compass necessary and the personal error in marking the glass tube and reading the goniometer.

Surveying in Magnetic Formations.

When the rock-formation is known to be magnetic, or when several tests with the compass do not agree, there seems to be no way of determining the course of a hole but by lowering the rods in such a way that the test-tube can be oriented at any point in the hole. We have done this by the method used by Mr. Deacon. The rods are first screwed together in one or two long lines on the surface, just as they will be lowered into the hole, with a bronze case at the end, and at intermediate points in the rods if desired, all the joints being made as tight as usual. Great care is necessary that no twist be left in the rods when screwing them together on the ground. This trouble is not experienced when there is snow, as the rods slip easily, and no torsion is introduced. When the ground is bare, difficulty may be avoided by placing level planks at short intervals for the line of rods to rest upon, and to prevent their touching the ground at any point. A straight stretch of railroad-track, with ties exposed, is very convenient for this purpose. If not more than 500 ft. long, the rods will turn on grass without leaving any twist in them.

When all connected, each joint, including the bronze case, is marked with a chisel, so that it can be screwed up again to exactly the same place. The joints are marked exactly on top as they lie on the ground, so that when the rods are in the hole the marks will point in the same direction. The rods are then broken into 20-ft. lengths, convenient for handling, being careful to keep one wrench on the couplings so as not to disturb any joints not to be broken.

Dilute hydrofluoric acid is poured into a glass tube, the tube marked with a diamond, and placed in the bronze case so that the mark on the tube corresponds with that on the case. A convenient way of doing this is to cut a mark across the top of the stopple, set this to agree with the mark on the tube, and then place the tube in the case so that the mark on the stopple agrees with that on the case. The tube is then lowered into the hole, being careful exactly to match the marks at every joint, and again keeping the wrench on the couplings. The mark on the last rod is placed directly in front of the drill and this direction determined, which is the direction of the mark on the glass tube. The tube is left stationary at the bottom of the hole for about 50 min. when acid diluted 12 to 1 is used, and then withdrawn and washed. To determine the course of the hole, another mark, exactly opposite, is made on the tube, and the course is found by using the goniometer as described in connection with the gelatine test.

This method, of course, gives accurate results only when the rods turn easily in the hole, so that there is no twist in them after they are lowered, which is usually the case except in very deep holes, or in holes where the inclination is low, or where the curvature is excessive. In these latter cases, unless the hole is rifled, the twist may probably be removed after the rods are in the hole by raising and lowering them several feet a few times. Precaution should be taken that the tube cannot turn in the bronze case, by forcing it in with a little waste alongside. On withdrawing the test it is well to examine each joint in the rods as they come out of the hole to see that none have moved while lowering or hoisting, and to be sure that the tube has not moved in the case.

This method requires great care and patience, but is capable of very good results. My experience has been that it cannot safely be left to assistants, whereas the compass method will give good results in the hands of any ordinarily careful and intelligent assistant, or even in the hands of the drillmen.

I have tested the same drill-hole by both methods and found the results to agree very closely. In one case the first tests in an old hole, by compass, did not agree very well because of casing left in the hole. Two or three tests were taken at each point, however, and the averages gave a smooth curve for the drill-hole. Later tests were made with marked rods, and the results agreed so well with the average of the compass-readings that when the two were plotted there was a difference of only 12 ft. at the bottom of the hole, although it was 1,697 ft. below and 262 ft. horizontally from the collar. The averages given below show that the hole described a spiral of considerable curvature. These results would have agreed even better if the compass had not been considerably affected by iron casing in the hole.

Depth. Feet.	Inclination. Degrees.	Course. By Compass. Degrees.	Course. By Marked Rods. Degrees.
100	89	N. 73 E.	N. 85 E.
300	87.75	S. 42 E.	S. 36 E.
500	86	S. 24 E.	S. 2 E.
700	86.5	S. 38 W.	S. 30 W.
900	85	S. 25 W.	S. 47 W.
1,100	79	S. 51 W.	S. 52 W.
1,300	71.5	S. 77 W.	S. 81 W.
1,500	65.75	N. 85 W.	N. 86 W.
1,700	64	S. 77 W.	No test.

Deflections of Drill-Holes.

J. S. Curtis[5] gives an interesting theory of the cause of bore-hole deflections, with results of experiments which he made to substantiate his theory. He endeavors to show that the influence of terrestrial magnetism should cause vertical drill-holes to deviate to the north in the southern hemisphere, and affirms that this is the case in the great majority of holes, although the direction may be changed by the character of the country-rock.

In our experience the latter feature is much the more important, and from results of our drilling I would not say that the great majority of drill-holes deviate either north or south in all districts. If the strata are flat and uniform the holes may do so, but if the strata dip deeply this is not the case. In one district, where the dip is steep, we are certain of the course of 14 holes which deviated from the vertical. Of these holes, one went approximately north, one approximately south, one NE., five NW., one SE., and five SW. Putting it in another way, seven deviated to the north and seven to the south, while two deviated to the east and ten to the west. If these results show anything, they only show that the majority of the holes deviated to the west, but equally to the NW. and SW.

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[5] *South African Mining Journal*, vol. viii., part 1, No. 379, p. 425 (June 11, 1910), and *Journal of the South African Institute of Mining Engineers*, vol. ix., No. 8, pp. 199 to 206 (Mar., 1911).

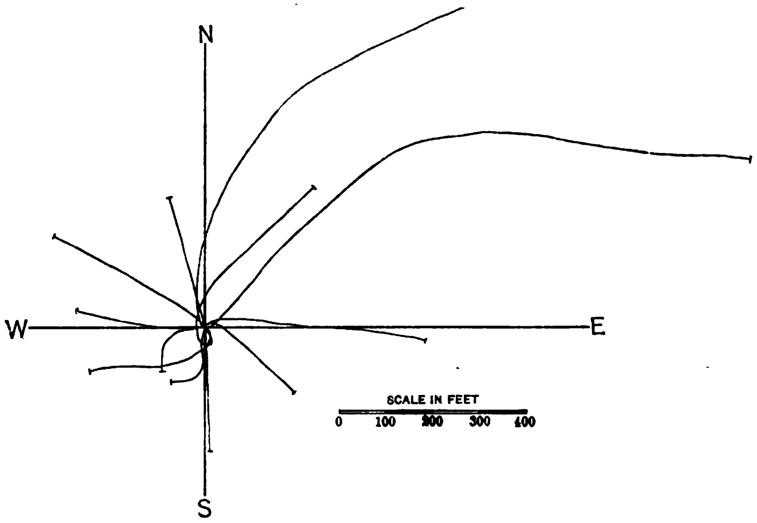

FIG. 7.—PLAN OF 12 DRILL-HOLES IN LAKE SUPERIOR DISTRICT.

Fig. 7 shows a plan of holes which deviated considerably from the vertical. These holes were drilled on several ranges in the Lake Superior district. It will be seen that there is no general course that can be predicted.

It is very difficult to keep vertical diamond-drill holes straight, and I believe that a hole can be located with more assurance of striking a certain point in depth if it is given an inclination of 85° against a steeply-dipping formation than if it is started in a vertical position. We have drilled only two holes with this inclination, the results being:

Depth. Feet.	No. 1. Degrees.	No. 2. Degrees.
At surface.	85	85.5
At ledge.	86	86.5
200	86	......
400	85	87.75
500	87.5	......
600	87	......
800	......	88.5
1,200	......	85.5

These two holes are not enough for a generalization, but they kept straighter than vertical holes in the same district. In addition, we knew in what general direction the holes would deviate, which we do not know when we start a vertical hole. In 1910[6] I gave a series of curves showing the curvature to be expected in an inclined hole when dipping against a steep jasper formation. I would change the curve for a hole started at 85°, since, under these conditions, that angle seems to be a critical one, and the hole does not flatten as would be expected.

In view of the sometimes surprising curvature of drill-holes, I feel that all holes should be tested both for course and for inclination at 100- or 200-ft. intervals, whether started vertical or at an angle, otherwise there is no certainty as to the exact place at which the ore or other strata is actually cut. We recently started a vertical hole which, at a depth of 800 ft., was found to have an inclination of only 51° from the horizontal. Another hole, started at an angle N. 54° W., was found to be running N. 64° E. at the bottom. A third hole, started vertical, deviated 377 ft. in a total depth of 1,290 ft., and was only 47° from the horizontal at the bottom.

Daily Reports.

Fig. 8 shows a report form filled out by the drillmen every shift for the drill-foreman and the head office, and Fig. 9 a sample-tag which is placed in every bag of core or sludge. The record of time of drilling and footage of each diamond-bit is kept, to obtain data on the several stones in the bit with the idea of determining which are the most economical. These tests have shown that the wearing quality of the stone depends considerably upon the specific gravity and upon the structure.

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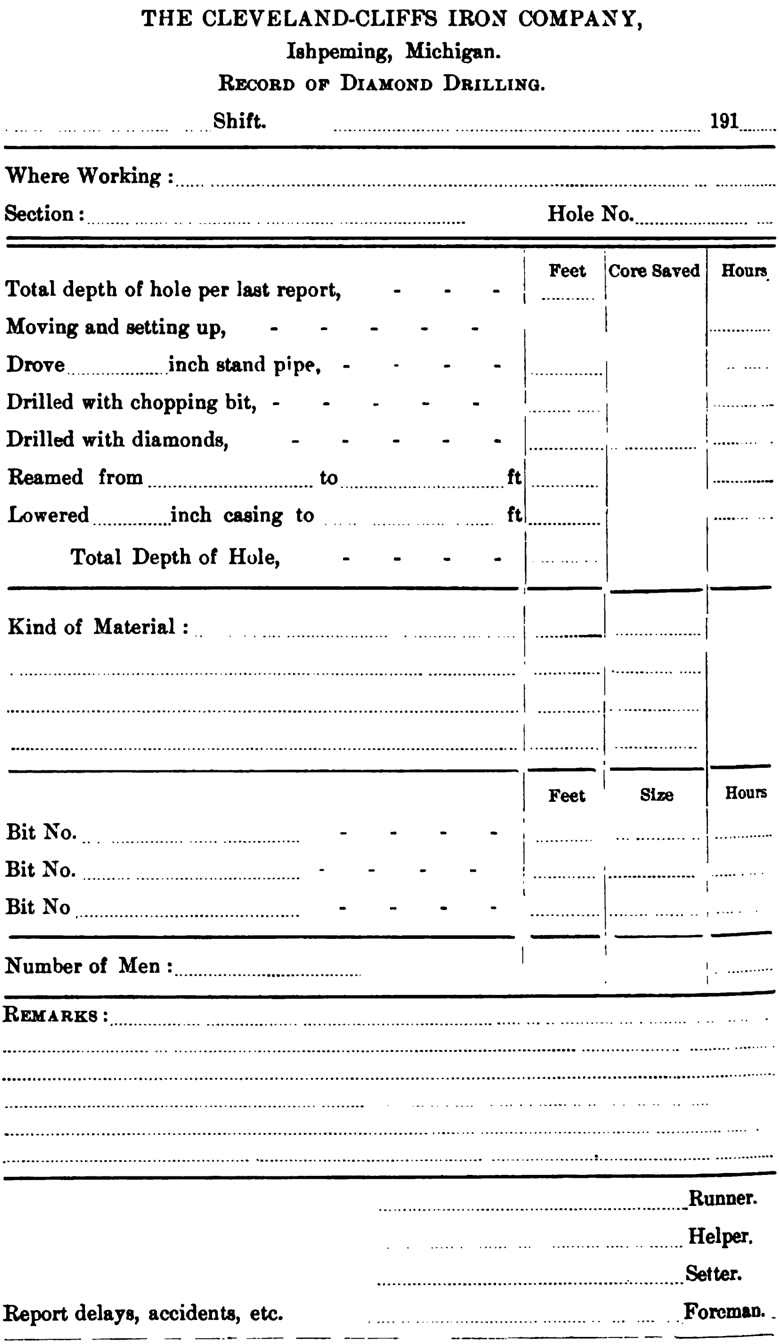
[6] *Engineering and Mining Journal*, vol. xc., No. 12, pp. 546, 547 (Sept. 17, 1910).



THE CLEVELAND-CLIFFS IRON COMPANY,
Ishpeming, Michigan.
RECORD OF DIAMOND DRILLING.

..........Shift. .... 191....

Where Working: ....

Section: .... Hole No. ....

	Feet	Core Saved	Hours
Total depth of hole per last report, - - - -	....		
Moving and setting up, - - - - - -			....
Drove .... inch stand pipe, - - - - -	....		....
Drilled with chopping bit, - - - - - -	....		....
Drilled with diamonds, - - - - - -	....	....	....
Reamed from .... to .... ft	....		....
Lowered .... inch casing to .... ft	....		....
Total Depth of Hole, - - - -	....		

Kind of Material:	....	....	
....	....	....	
....	....	....	
....	....	....	

	Feet	Size	Hours
Bit No. .... - - - -	....	....	....
Bit No. .... - - - - -	....	....	....
Bit No .... - - - -	....	....	....

Number of Men: ....

REMARKS: ....

....

....

....

....

....

.... Runner.

.... Helper.

.... Setter.

Report delays, accidents, etc. .... Foreman.

FIG. 8.—REPORT FORM FOR RECORD OF DIAMOND-DRILLING.

(Original size, 5 by 8.5 in.)

District..

Sec.................... Hole No. ....

Depth,..

FIG. 9.—FORM OF TAG FOR SAMPLES.
(Original size, 2.5 by 3.5 in.)

Although it is important to have the drillmen report the amount of core saved from each material, yet they rarely measure it accurately, and if the analyses of core and sludge are to be combined, as described below, the core is remeasured when it reaches the office.

Samples from Drill-Holes.

The following directions for saving samples are posted in the drill-shanties and enforced by the inspector:

Directions for Saving Sludge from Diamond-Drill Holes.

Set the standard sludge box just below the floor of the shanty and in such position that there is room to siphon off the water and take out the sample without moving the box. Connect a tee to the top of the standpipe or casing and lead a pipe from it to the nearer end of the sludge box, at such a height that it will either be level or slant towards the sludge box and just rest upon the top of the box, and of such a length that it will not project more than one inch beyond the edge of the box. The pipe must not be more than two feet long, and if longer than one foot must be split on top for the foot nearest the sludge box, so that if sludge collects in the pipe it may be seen. Set the box level, so that water will overflow evenly across the whole width at the far end, and wedge the partition firmly, so that it is in close contact with the bottom of the box. The top of the partition should be one inch below the water level. The box is now ready to receive the sample, and drilling may be started.

While drilling, care must be taken that no water from the drill-hole escapes around or over the tee except through the pipe leading to the sludge box. Care must also be taken that there is no leak from the box and that the three-inch plug at the end of the box is tight. Sludge samples must be taken for every five feet drilled, or less, preferably from even five-foot intervals; that is, from 460 to 465, 465 to 470, 470 to 475, etc.

Whenever a sludge sample is to be taken, drilling must be stopped and the hole washed out clean. The pipe leading to the sludge box must be cleaned out into the sludge box, and either the pump must then be stopped or the tee turned so that the water will not be discharged into the box. Carefully remove the partition in the box, so as not to stir up the sludge any more than necessary, and when the sludge is settled, siphon off the surplus water, being careful to keep the end of the siphon near the surface of the water, and not disturb or draw off any of the fine sludge at the bottom of the box. To use the siphon, fill with water a three-foot length of large size flexible hose, and with one hand on each end place one end beneath the surface of the water in the box and the other end on the ground eight inches or more below the top of the box. When both ends of the hose are released the water will flow out of the box, and may be allowed to flow until it is seen that the sludge is beginning to go off with the water. Then remove the hose and thoroughly mix the sludge in the box to a mud. This must all be removed from the box and placed in a pan on the boiler to dry. The pan must be at least 8 in. by 12 in. by 1 in. deep, with flat bottom, and must be thoroughly cleaned each time before a sample is put in it to dry. If enough water cannot be drawn off without disturbing the sludge so that the sample can be contained in this pan, a larger pan must be used. All the sludge must be saved and the sludge box cleaned thoroughly. When the sludge has been cleaned out, remove the three-inch plug at the end of the box, and wash out the box with a pail or two of water, then replace the plug and partition, and drilling may be started again. The sludge must be labeled, giving the depths between which the sample was taken, when it is placed on the boiler to dry. It must all be saved and turned over to the inspector. Sludge must always be saved when drilling in iron formation or in any other ferruginous or red material. While drilling in material from which a sludge sample should be saved, if the water is lost, if the sludge does not come up with the water, or if the sludge is contaminated with material caving from higher up in the hole, drilling must be stopped immediately until the hole is put in such condition that good sludge samples can again be obtained, or until the inspector gives orders that drilling may proceed.

Whenever the drill runs into or out of ore, provided the band of ore or rock is one foot or more thick, drilling must be stopped, and the sludge box cleaned out immediately, without waiting to complete the five-foot run. When the drill runs out of ore continue taking and saving sludge samples for at least twenty feet, no matter what the material, so that it may be determined whether the ore is caving.

Keep the core separate from the sludge, and each time core is pulled, label it with the depths between which it was recovered. Each run of core must be kept separate, and all core must be saved and turned over to the inspector. When the core is pulled, if it is found that more core is saved than the proportion of one foot of core to ten feet of drilling, the sludge box must be cleaned without waiting to complete the five-foot run, and the sludge labeled and saved separately. If sludge from a shorter distance than five feet is in the box at the end of the shift's work, and if less than the above proportion of core is saved, the sludge may be left in the box, provided the shanty is locked and the box is inaccessible from outside the shanty. If anybody can get at the box, however, and if there is no watchman, the sludge must be removed from the box, dried, labeled, and placed with the other samples.

The standard sludge-box is shown in Fig. 10.

When the samples reach the office they are carefully examined, and a daily report of all drilling made out on the form shown in Fig. 11. Samples of all core and of all sludge which runs above 40 per cent. of iron are preserved in special cabinets in a fire-proof room. A few pieces of core are saved from each run, and the rest sent to the laboratory for analysis if the ore-formation has been cut, otherwise it is thrown away. A little of each sludge-sample is placed in a small pasteboard tray, with a temporary label, 10 ft. to a tray, until the analysis is completed. Each 10 ft. of sludge-sample which exceeds 40 per cent. of iron is then placed in a gelatine case 3 in. long by 1 in. in diameter, and preserved in the same drawer with the core. Gummed paper labels are used for both core and sludge.

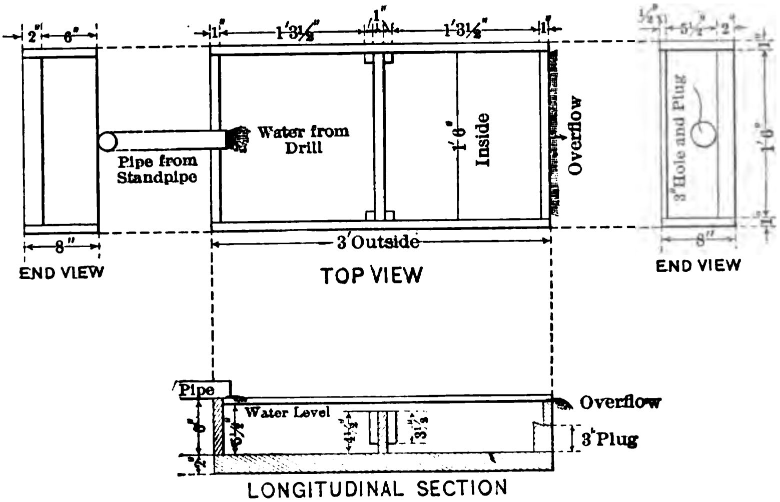


FIG. 10.—CLEVELAND-CLIFFS IRON CO. STANDARD SLUDGE-BOX.

THE CLEVELAND-CLIFFS IRON CO.

DAILY REPORT OF DIAMOND DRILLS.

Ishpeming, Mich., 191

Section	Hole No.	Date	Feet Drilled	Total Depth.	Material	Remarks

FIG. 11—FORM FOR DAILY REPORT OF DIAMOND-DRILLS.

(Original size. 8.5 by 11 in.)

Combination of Core- and Sludge-Analyses.

When enough core is saved to make it worth while, and when the analyses of the core or sludge exceed 45 per cent. of iron, the results are combined according to a formula which gives proper weight to each, derived as follows:

Let A = diameter of bit outside of carbon, in inches.
B = diameter of bit inside of carbon, in inches.
C = feet of core saved in "D" feet drilled.
D = feet drilled.
S = volume of rock actually ground to sludge, in cubic inches.
T = volume of rock actually saved as core, in cubic inches.

$$\text{Then } S = 12\,D\left(\frac{\pi A^2}{4} - \frac{\pi B^2}{4}\right) + \left(D - C\right)\frac{\pi B^2}{4}$$

$$\text{and } T = 12C\,\frac{\pi B^2}{4}$$

$$\text{Hence } \frac{S}{T} = \frac{D(A^2 - B^2) + (D - C)B^2}{CB^2} = \frac{DA^2}{CB^2} - 1$$

	Dimensions of Bits. Inside of Carbon.	Outside of Carbon.
Standard A bit	1 in.	1 13/16 in.
Standard E bit	27/32 in.	1 9/16 in.

$$\text{Hence for an A bit, } \frac{S}{T} = \frac{1.813^2 D}{C} - 1 = 3.29\,\frac{D}{C} - 1$$

$$\text{and for an E bit, } \frac{S}{T} = \frac{1.563^2 D}{0.844^2 C} - 1 = 3.43\,\frac{D}{C} - 1$$

To obtain an average of the sludge- and core-analyses, giving the proper weight to each, the sludge-analysis should evidently be multiplied by S, and the core-analysis by T, and the sum of the products divided by $S + T$. The result is the same and the operation simpler to multiply the sludge by $\frac{S}{T}$ and the core by 1, and divide by $\frac{S}{T} + 1$; hence, the rule for an A bit is to multiply the sludge-analysis by $3.3\,\frac{D}{C} - 1$, add the core-analysis, and divide the sum by $3.3\,\frac{D}{C}$.

Only iron is run on the 5-ft. samples, with phosphorus if they contain more than 40 per cent. of iron, and manganese if exceeding 1 or 2 per cent. Occasionally, several of the 5-ft. samples are combined, and a complete analysis made for iron, phosphorus, silica, lime, magnesia, alumina, manganese, sulphur, titanium, and loss by ignition.

After combining the core- and sludge-analyses, the results are further averaged in continuous runs of ore of the same grade. We call from 45 to 50 per cent. of iron "lean ore," 50 to 57 per cent. "second-class ore," and more than 57 per cent. "first-class ore."

Representation of Results.

As soon as possible after the first of the month, the record of material drilled during the previous month is compiled from daily reports and averaged analyses. This result is carefully checked with the core and then recorded permanently in the drill-book, which has loose leaves, with pages of the form shown in Figs. 12 and 13. The reports of the drillmen and analyses of samples are copied in this book daily, and it forms the complete and permanent record of drilling. From this book tracings are plotted from which blue-prints are made. Figs. 14 and 15 show the printed forms on tracing-cloth used for this purpose. The tracings are 14 by 16 in. in size, the same as that of the loose-leaf sheets in the drill-book, and are bound in covers of the same size.

Cross-Sections of Drilling.

For the scientific location of drill-holes, cross-sections must be made through outcrops and previous drill-holes, preferably at right angles to the strike of the formation, showing the holes plotted according to the surveys for inclination and course, and showing the material encountered. We make these on cross-section tracing-cloth ruled in inches and tenths, on a scale of 50 ft. to 1 in. The sections may be superposed and compared, and the cross-section ruling makes it easy to read distances and areas without a scale. On these tracings the geological boundaries of formations and the outlines of ore-bodies are drawn in soft pencil, which prints satisfactorily, but may easily be erased and changed if further drilling shows the first assumptions to be wrong. When necessary, longitudinal sections are made, which, in connection with a plan and cross-sections, give a very good idea of the structure of the deposit.

THE CLEVELAND-CLIFFS IRON CO.

RECORD OF DIAMOND DRILLING.

District } ....

State ...... *T.* ... *R.* ...... *Sec.* ...... *Hole No.* ...... *Location:* ......

Mine } ....

Dip: .... *Course:* .... *Drilled by whom* .... *with* ....

Date	No. Hours	Core Recovered		Stand Pipe	Drilled	Total Depth	Kind of Material	Begins at	Dip of Strata	Remarks
		Ft.	In.							

FIG. 12.—LOOSE-LEAF PAGE FOR RECORDS OF DIAMOND-DRILLING.

(Original size, 14 by 16 in.)

THE CLEVELAND-CLIFFS IRON CO.

District ....

ANALYSIS OF CUTTINGS AND CORE.

Mine ....

State ...... *T.* ...... *R.* ...... *Sec.* ...... *Hole No.* ...... *Location:* ......

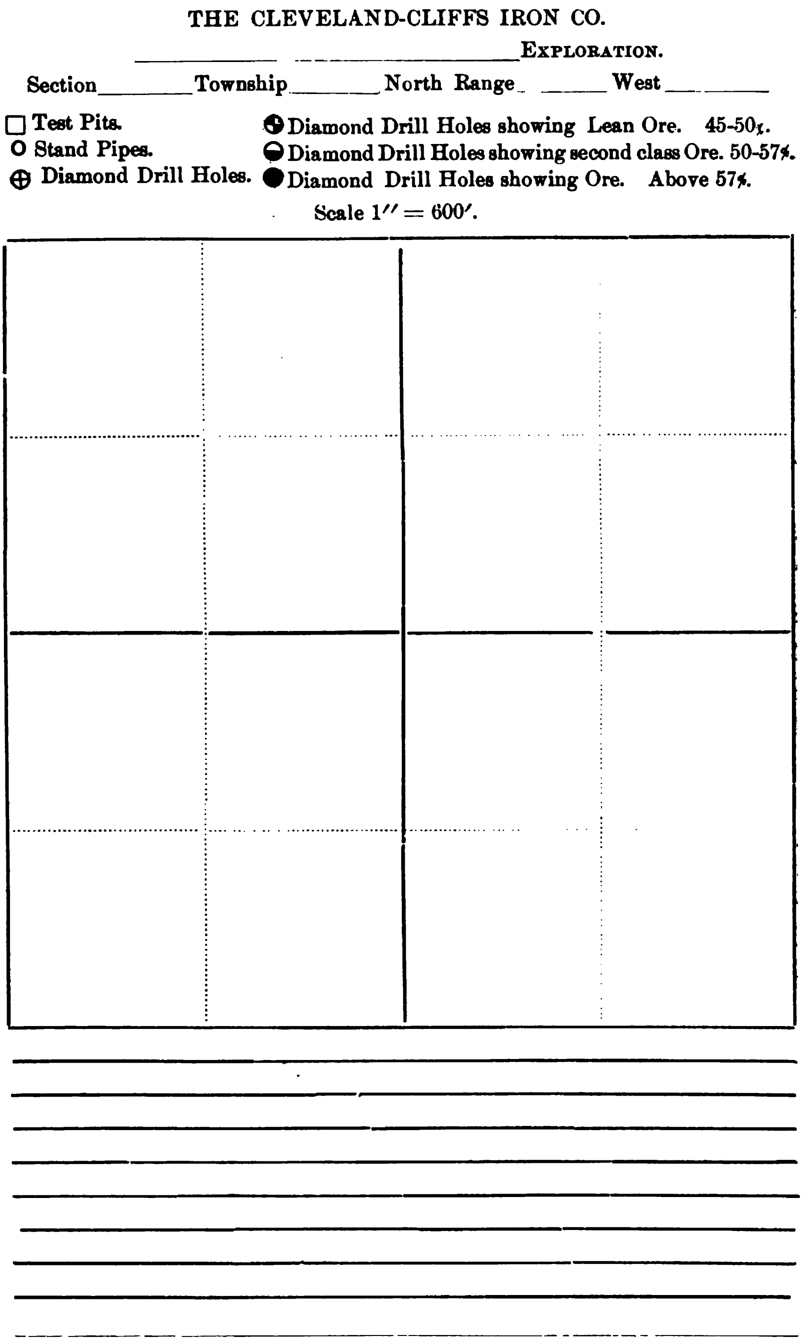
Option ....

Lease .... *Determined by* .... 191 ... *Elevation of Collar* ....

From	To	Iron	Phos.

FIG. 13.—LOOSE-LEAF PAGE FOR RECORD OF ANALYSES.

(Original size, 14 by 16 in.)



THE CLEVELAND-CLIFFS IRON CO.

__________ ________________EXPLORATION.

Section________Township________North Range_ ______ West_________

☐ Test Pits. ◕ Diamond Drill Holes showing Lean Ore. 45-50%.

O Stand Pipes. ◓ Diamond Drill Holes showing second class Ore. 50-57%.

⊕ Diamond Drill Holes. ● Diamond Drill Holes showing Ore. Above 57%.

Scale 1″ = 600′.

FIG. 14.—LOOSE-LEAF TRACING-CLOTH FORM FOR PLAN OF EXPLORATIONS. Original size, 14 by 16 in.)

THE CLEVELAND-CLIFFS IRON CO.

______ ___ ___ _______ Exploration Diamond Drill Holes.

Section ________ Township _________ North Range _________ West _______

Scale 1″ = 50′.

Fig. 15.—Loose-Leaf Tracing-Cloth Form for Sections of Drill-Holes.
(Original size, 14 by 16 in.)

SUBJECT TO REVISION.

[TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS.]

The Manufacture of Coke.

BY WILLIAM HUTTON BLAUVELT, SYRACUSE, N. Y.

(Cleveland Meeting, October, 1912.)

COKE is the product of dry distillation of bituminous coal, by which the volatile matter is driven off, producing a hard body of cellular structure. Not all bituminous coals will coke, and there has been much discussion regarding the substance which is present in coking-coals to distinguish them from dry coals. Professor Lewes says that the bituminous matter in coal is largely derived from the spores of fossil mosses. Resins are present in all soft coals, and contribute largely to the cementing of the coke. The resinates and hydrocarbons begin to distil out at moderate temperatures and leave the pitchy residual, which at 500° C. forms a mass already well caked together, if the proportion of humus residual is not too great. This coke is soft, but if the heat be raised to 1,000° C. the pitch decomposes further, leaving carbon, and Professor Lewes is convinced that the coking bond is due to these liquid products which distil off, leaving pitch, which then carbonizes and binds the mass into coke.

Professor Parr concludes that the structures of the organic compounds of the coal which furnish the cementing-material for coke, and which are apparently attacked by oxygen, have not been determined and seem to vary somewhat in different types of coal. However, they yield, on oxidation, humic acids of varying composition which decompose into powdery residues. Because of the complex nature of these substances and the difficulty experienced in isolating and identifying them, the theory of coking is still an open problem and the explanations advanced are largely hypothetical.

In the coking process the volatile matter driven off in the oven consists mainly of tar, ammonia, and gas, familiarly known as by-products. Both the quality and quantity of coke and by-products vary greatly with the composition of the coal.

In America, coals similar to those of the Pocahontas region, containing as low as 16 per cent. or less of volatile matter, stand at one end of the list. In Europe, some coals are coked which contain not more than 13 per cent. of volatile matter. These produce the maximum yield of coke and the minimum yield of by-products. At the other end of the list are the gas-coals, containing as much as 38 or 40 per cent. of volatile matter, and yielding correspondingly small amounts of coke. The development of the best cell structure is of the highest importance in the production of metallurgical coke. A well-developed cellular structure presents a large surface to oxidation by air, producing rapid combustion, while the strong cell-wall prevents crushing and maintains an open fuel-bed. A soft thick cell-wall is objectionable in blast-furnace coke, since it permits the oxidizing action of CO_2 gases in the upper part of the furnace, by which the coke is wasted and the furnace-top temperature is increased. In foundry-work it is more desirable to have a coke with a softer cell-wall, smaller cell space, and a strong, tough structure. This permits the air to burn the coke immediately to carbon dioxide, with production of the highest temperature in the melting-zone of the cupola; and the weight of the charge is easily supported. Thus present practice seems to indicate that somewhat different treatments are required for producing furnace- and foundry-coke.

Coals high in oxygen are usually deficient in coking-qualities, and either will not coke at all, or produce a coke of weak, friable structure. But many coals, as will be shown later, which yield poor coke under ordinary conditions, may, by proper treatment, be made to produce good metallurgical coke. Coal when stored absorbs oxygen, sometimes with serious effect on its coking-qualities. The chief reason for this appears to be the effect of the oxygen on the resinates, which Professor Lewes says are thereby converted into humus.

The history of coke making has been discussed at length in the technical publications, by me[1] and others, and any reference to it here would be unnecessary repetition. The same statement applies to the earlier forms of bee-hive and by-product ovens, and the development from these earlier forms to modern types. The term by-product oven is commonly applied in this country to the retort oven, although, of course, the by-products may or may not be recovered from the gases from retort ovens, depending upon market conditions. All the retort oven-plants in the United States have by-product recovery-apparatus. In this paper, the term by-product oven will often be employed, as being a more familiar term, in referring to the retort oven.

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[1] *Mineral Industry*, vol. iv., p. 215 (1895).

For many years the bee-hive oven was the standard for coke making, and has remained the standard in England and America long after its practical disappearance from continental Europe. There are several reasons for this. In England, the Durham coal, and in America, the Connellsville coal, are remarkably well-suited to the conditions of the bee-hive oven, so that the best quality of coke could be produced without difficulty, and with relatively good economy. In America, the bee-hive oven had certain economic advantages. It is quickly built, and at relatively low cost, and the labor required for operation is of low grade. It can be put out of run at relatively small loss during periods of industrial depression, and can be started up again with ease after a shutdown. These qualities adapted it to the fluctuating and changing conditions which for many years surrounded our iron industry. But the iron industry has become more stable; and moreover, the coals best adapted to the bee-hive oven are growing scarcer, especially with the exhaustion of the Connellsville field. The U. S. Geological Survey reports that in 1911 the total coke production in the United States showed a decrease of about 15 per cent. while the output of by-product coke increased 10 per cent. Compared with five years ago the following are the figures of production:

	By-Product.	Bee-Hive.
1907	5,607,899	35,171,665
1911	7,847,845	27,207,517

This shows an increase in by-product coke of 40 per cent., and a decrease in bee-hive coke of 22 per cent., so that in 1911 the tonnage of by-product coke was 28 per cent. of the bee-hive tonnage. By-product plants now under construction and in contemplation will greatly increase the present output. In England the relative growth of the by-product oven has been even more rapid during the last few years.

The generic differences between the structure of the bee-hive and by-product or retort oven need not be again described here. It is sufficient to say that the bee-hive process consists essentially in the heating of the coal with controlled admission of air to the coking-chamber, so that the heat necessary for the distillation of the volatile matter is produced by combustion within the oven-chamber. In the by-product oven the process is a true dry distillation. No air is admitted to the chamber, and the heat necessary for the distillation is supplied through the chamber walls.

The bee-hive oven has been carried to its highest perfection in this country; drawing- and loading-apparatus has been applied to it, to reduce the arduous labor of the old hand-method; and modifications of the original bee-hive shape have been successfully introduced, which add materially to the economy of operation, while maintaining the principles of carbonization as employed in the standard bee-hive.

The bee-hive oven is usually located at the coal-mine, and the coal is in very many cases charged direct to the oven without weighing, so that it is difficult to ascertain the exact yield of the oven; but except when coals are used which are especially adapted to the bee-hive process, the coke yields are considerably below the theoretical, and, of course, all the by-products are wasted. These facts, except in unusual cases, make it impracticable to locate bee-hive ovens away from the mines. On the other hand, the by-product oven is usually, in this country at least, located at the point of consumption of the coke, or at some center of distribution. This freedom of location of the by-product oven has a number of advantages. Although it entails freight charges on from 1.2 to 1.4 tons of coal for every ton of coke produced, yet coal usually carries a lower rate than coke, and is more easily transported, not requiring special cars and not being injured by handling. In some cases it can be shipped by water, with material saving in freights, and with proper care it can be stored at the plant in almost any quantity without material deterioration. This permits a blast-furnace plant, having its own coke-ovens at the furnace, to possess an assured supply of coke, independent of weather or shipping-conditions. It is quite common for by-product oven-plants to accumulate a stock of from one to eight months' supply of coal. The by-products produced are much nearer their market, and the gas is often available for industrial uses or for municipal lighting. The plant is nearer a supply of diversified labor, which is an advantage in the more varied processes of by-product oven-operation. An important advantage of locating the oven-plant at the point of consumption is, that it permits a convenient assembling of several kinds of coal at the ovens. This mixture of coals is often a great advantage, since it permits the best quality of coke to be produced, when the coke made from any one of the coals alone might be of inferior quality, or perhaps not at all adapted to the market requirements.

There has been much discussion regarding the relative cost of bee-hive and by-product oven-plants. It is difficult to make an exact comparison, since the functions of the two types are quite different. For example, in many cases the cost of a by-product oven-plant includes a large expenditure for coal-storage of several hundred thousand tons, in order to take advantage of water freight-rates. On the other hand, bee-hive plants are often built in connection with coal-mining plants and utilize a share of power-plant, water supply, etc., without having these necessary adjuncts included in their capital account. Careful inquiry into the actual cost of bee-hive oven-plants shows that to build such a plant complete in every respect, and in the best manner, including all the equipment besides the ovens and their immediate appurtenances, such as electric power-plant, water-supply, railroad-approaches and sidings, coal-handling, etc., would require an expenditure of about $950 per oven. The U. S. Geological Survey report for 1911 gives the tonnage of coke produced per active bee-hive oven at 466 tons per annum. This figure is doubtless low for the best bee-hive ovens; 675 to 700 tons per annum will perhaps represent fairly the average output per oven of a modern bee-hive plant. An average of these figures gives a plant-cost of $1.38 per ton of coke produced per annum. A by-product oven-plant of, say, 80 ovens, complete in every respect, and built in the best manner, would produce, say, 425,000 tons of coke per annum from an average coal, and would cost, say, $1,100,000. Of course, this figure would be varied by local conditions. This is equal to $2.58 per ton of coke per annum. On the basis of these figures a by-product plant costs 86 per cent. more than a bee-hive plant.

The owner of a small acreage of coking-coal might perhaps well hesitate before making the larger investment, as compared with a simple plant of bee-hive ovens, but, as a general proposition, there hardly seems to be a question as to which style of oven is the better investment. Much of the additional cost of the by-product plant per unit of coke produced is, of course, due to the installation of the apparatus and buildings for the recovery and treatment of the by-products. Other important items are provisions for coal- and coke-storage, thereby assuring uniformity of operation and greater ability to maintain uniform deliveries. Moreover, a by-product plant is built for a longer life than a bee-hive plant. Eight years may perhaps be considered as a satisfactory life for a bee-hive oven-plant, while double this term, or 16 years, would be several years within the life of a well-built by-product plant.

Market conditions have not been such in this country as to justify construction of retort oven-plants without the recovery of by-products, but we may assume a non-recovery oven-plant built for operation in the Pocahontas region, for example. The yield of coke from Pocahontas coal averages certainly not more than 60 per cent. in a bee-hive oven. In a retort oven the yield is more than 80 per cent. Such a plant without by-product recovery costs not much more per ton of product than an equally well equipped bee-hive plant, and the labor of operation under such conditions would be as low, if not lower. A plant of retort ovens in place of bee-hives in that field would, therefore, mean that at practically the same expenditure the owner of a coal property would, on account of the greater yield of coke, increase the life of his coal-field 33 per cent. with a given output of coke. Or the value of the property, based on the selling-value of the coke therefrom, would be increased 33 per cent.

The present development of the art, however, has brought the Pocahontas coal to the ovens located at the point of consumption, and the mixing of low and high volatile coals has grown in favor, especially during the last few years.

The cycle of operations in connection with the by-product oven has been described several times in technical papers, and need not be repeated here. It is well recognized that the operation of these plants calls for technical skill of high order, as the several processes include metallurgical, mechanical, and chemical engineering, and the success and economy of the operation are largely dependent upon the accurate control of a number of quite different and independent conditions. There has been great progress in the design of the ovens and recovery-apparatus. As might be expected, America now leads Europe in output of plant, size of oven, and rate of coking. In the early days of the by-product oven in America, the capacity of the oven was about 4.4 tons of coal per 24 hr., and 25 ovens were considered about the right number for one crew of men. Modern ovens have a capacity of as much as 20 tons of coal per oven per day, and by the introduction of more machinery and more efficient design, the number of ovens handled per man is also increased greatly. The rate of coking is one point in which American practice has gone ahead of Europe.

Retort ovens have been built of various widths from about 14 to about 30 in., to suit various ideas of the designers, and various coals; but, within limits, the rate of coking per inch of coal does not vary materially with the width of the oven. Not many years ago the best rate of coking was about 1 in. of oven width in 90 min.; that is, a 16-in. oven was coked in 24 hr. To-day there is more than one type of oven which is coking regularly at the rate of from 50 to 55 min. per inch of oven width. This increased rate was made possible, partly by better control of the heating-systems, and partly by the adoption of silica brick in the oven construction. Silica brick has been used in bee-hive oven construction very generally for a number of years. In retort ovens it was first used in the Otto-Hoffmann oven-plant at Johnstown in 1899, and is now the standard material in America for retort-oven construction.

During the past few years special study has been given to the heat economy of the retort oven and material advance has been made. From most American coals, more gas is produced than is necessary for heating the ovens. The conditions under which it is to be operated have an important bearing on the design of an oven-plant. It may be said that there are three general conditions. First, where there is not sufficient market to dispose of the surplus gas. In this case the design of oven should be very simple. Any complication of the design for preheating the air should be avoided, since satisfactory coking-temperatures can be maintained with cold air, if economy of gas is no object. Steam for the operation of the plant should be raised by burning the surplus gas under simple boilers. This simplicity of design permits material economy in cost of construction and maintenance, and also reduces the cost of operation.

The second condition is where there is a market for the gas, perhaps for industrial purposes, but at a low price, and the cost of coal and labor is relatively high. In this case the oven-design should provide for partial heating of the air, with proportionate economy of gas for heating the oven, so that the remaining heat in the waste gases is just about sufficient to furnish the steam required for the plant. This design gives an oven of moderate cost for construction and upkeep; the operation of the boiler-plant is reduced to its lowest terms, with minimum of labor and repairs; and at the same time a very considerable percentage of the total gas produced is available for sale.

In the third case the gas is salable at a good price, so that it is desirable to recover the maximum amount for sale. Under these conditions an oven-design is justifiable which, while more costly in construction, is most economical in heat-consumption. The air is heated in the best form of recuperators or regenerators to 1,000° or 1,100° C., and the waste gases go to the chimney too cool to be of value. Under these conditions coal- and coke-breeze are used for producing the necessary steam. Whether steam is used for the production of power, as well as for the distillation of ammonia, heating, etc., or whether the power is generated by gas-engines, depends upon the relative market value of coal and gas. A proper recognition of the conditions of installation, and the adaptation of the design to them, is an important factor in the most economical installation of a retort oven-plant. It is by no means always the case that the most highly efficient heat-economy within the oven itself is most economical for the plant, and very good results can be obtained in some cases without the most expensive oven-construction. The average of a year's results from two plants which have come within my knowledge, using coal containing less than 27 per cent. volatile, showed more than 4,200 ft. of gas sold per ton of coal coked for the entire year. Statements regarding volumes of gas recovered are often misleading, since the quality may vary greatly: so it may be of more interest to say that the heat-units in this gas averaged from the two plants for the entire year well over 2,600,000 B.t.u. per net ton of coal coked, or about 44 per cent. of the heat in the total gas. While these figures can be materially improved by more expensive construction to effect greater heat-economy in the oven, the above results were obtained with the simplest form of apparatus for preheating the air, and at the same time all the steam required for operating the plant, handling and storing coal, distilling ammonia, etc., was produced from the waste heat, assisted by the breeze produced at the plant.

The following tabulated data show in a general way what may be expected from plants designed for the three conditions mentioned above. The figures show the results per ton of coal coked. They represent average conditions and average coals, and, of course, would be modified by special conditions.

Type of Oven.	Fuel Gas. Per Cent.	Surplus Gas. Per Cent.	Steam Produced. Pounds.
No air preheating	70	30	1,050
Partial air preheating	60	40	800
Maximum air prehea	40	60	0

The tendency of modern retort-construction, especially in America, is towards the highest economy of heat, even at the expense of simplicity and economy of construction. The proper heating of a retort oven is not a simple problem. It consists essentially in the distribution of heat from burning gases over the side wall of the oven, which presents an area from 35 to 40 ft. long, and from 8 to 12 ft. high. The temperature over this entire area must be always under accurate control. When operating for maximum output the temperatures must be held at a point not far below the softening-point of the brick; and since the oven-chamber is wider at one end than at the other, to permit easy discharge of the coke, the heats must be modified accordingly, so that the whole charge may be coked in the same time. It is also often desirable to maintain a somewhat lower temperature in the upper part of the oven. The combustion of the gas with air which has been preheated to, say, 1,100° C., as it is in the latest regenerator ovens, produces a theoretical flame-temperature of about 2,400° C. The actual temperature produced would be much above the melting-point of the best refractory brick, and while economical operation demands a full utilization of the best temperatures attainable, the walls of the combustion flues must not be injured, nor must there be "hot spots" or "white ends" that will overburn the coke in the adjacent part of the oven-chamber.

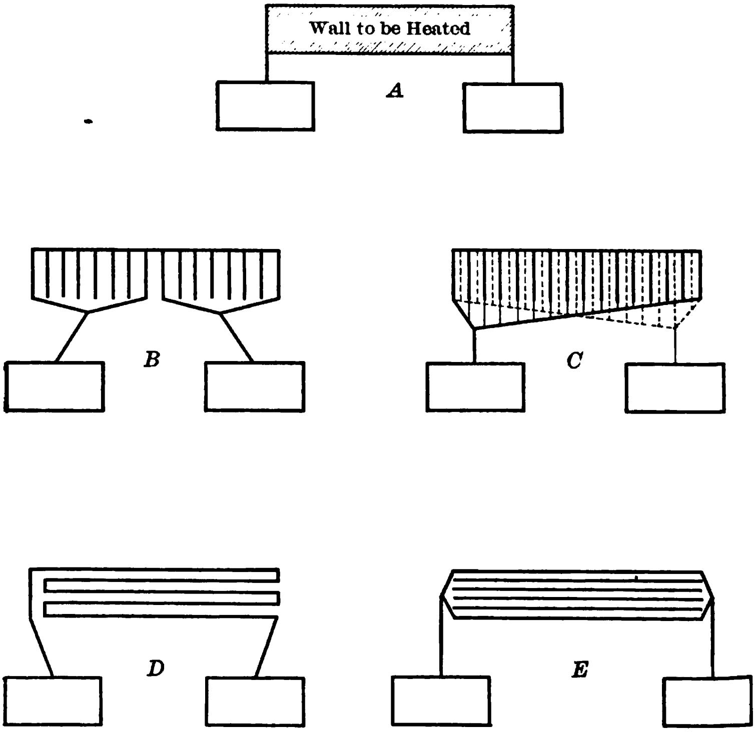

FIG. 1.—APPLICATION OF HEAT IN DIFFERENT TYPES OF COKE-OVENS.

Fig. 1 shows diagrammatically the application of the heat in the different types of ovens. *A* indicates the combustion-flue system of any retort oven with its regenerative chambers below. *B* and *C* show applications of this general arrangement as adopted by oven-systems employing vertical flues. *D* shows the application to the horizontal-flue ovens of the series type. *E* a horizontal type where all the flues are in parallel. *B*, *C*, and *D* represent the methods generally adopted in commercial ovens. In the former type, or the vertical-flue oven, the gas and air are mingled and burned in horizontal flues at the bottom of the vertical flues, the burning gases distributing themselves and passing up one-half the vertical flues and down the other half, as indicated in the sketch; or gas and air are led separately to the bottom of each of the vertical flues, where they are mingled by properly arranged nozzles and burned as they pass upward. In some ovens a combination of these methods is adopted. *D* illustrates the horizontal series-flue regenerator oven; and, as I have had more experience with this type, it may be of interest to discuss somewhat in detail the principles of its operation. All of the air required for the combustion is passed through the entire flue-system, first from above downward, and then from below upward, as the regenerator system is reversed. The heating gas is admitted at the end of the flues, usually in four or five places, as may be required. This method seems to give a maximum of simplicity. The flow of gas is automatically reversed in the flues themselves by the change in the current of air. Every flue may be conveniently inspected from end to end by a man walking along in front of the ovens, and the gas-admission at each point is directly under his hand, so that there is no excuse for failure to immediately observe and correct any tendency to unequal heating. The distribution of the heat throughout the flue-system is made remarkably uniform by this process. This is partly due to the comparatively high velocity of the gases sweeping through the flues, and to the fact that the gas is burned either in a large excess of highly-heated air, or else in a mixture of air and products of combustion. The advantages of the presence of products of combustion in the combustion-chamber, where it is desired to obtain a distribution of heat, have been clearly shown in other combustion processes, such as the Doherty system of producer operation, for example.

In beginning the experimental work leading up to the present method of operation, while it was recognized that the presence of a large volume of air, together with products of combustion mixed with the burning gas, was an important factor in distributing the heat with uniformity, yet it was feared that passing the entire volume of air through all the flue-system might seriously increase the friction and make the maintenance of sufficient draft a difficult matter, even though the total amount of air admitted could be accurately proportioned so that it would be just sufficient to burn all the gas. Hence, an oven was designed with the flues in parallel, in order to reduce the velocity of the gas-currents, following the principle of the vertical-flue type. It was found, however, that the velocity of the gases passing through the flues did not produce material friction-loss, but that most of the friction arose from eddies in the currents at the ends of the flues; and by a suitable modification of the structure at these points it was easy to keep down the draft required to a reasonable amount, while maintaining the velocity of flow that was so effective in securing uniform heats. This method of handling the gases through the flues in these series-flue ovens has, therefore, two advantages. It is effective in distributing the heat uniformly from one end of the flue to the other, and at the same time the higher velocities effect a more efficient transfer of heat through the flue-wall. The effect of velocity of travel on the transfer of heat through walls has also been shown very clearly in a series of experiments on steam-boilers made by the Engineering Experiment Station of the University of Illinois.[2] It will be remembered that these tests were undertaken to determine the relation of heat-transmission to velocity of circulation in the steam-boiler. They show very clearly the important effect of velocity flow on rate of heat-transmission, and from the deduction therein set forth it would appear that the same conclusions may be drawn in the study of our problem. Having obtained by these means such an effective distribution of the heat, the arrangement of gas-admission to the flue-system makes it easy to control the temperature of the whole system under the severe conditions described above. The result is that in this type of oven the heat is remarkably uniform from end to end, hot or cold spots are practically unknown, and the control of the temperature is much easier than ever before.

The temperatures maintained in the flue-system, depend, of course, upon the rate of coking it is desired to maintain. They usually run from 1,000° to 1,250° C., or somewhat higher. The temperature at the point of entrance of the heated air is controlled by the admission of the proper supply of gas, and additional gas is introduced at other points in the system in order to make up for the heat transmitted into the oven-chamber, or to augment the temperatures as may be necessary in preserving the proper relations of heat in the different parts of the system. The periodical reversal of the gas-currents prevents any tendency to inequalities which might otherwise develop.

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[2] *Bulletin No.* 40, *University of Illinois* (1909).

The perfecting of this system makes the regenerative retort oven quite as simple and easy to operate as the recuperative oven, and at the same time secures the maximum heat-economy. There is only one reversing-valve on each oven-block, by which the currents of air and products of combustion are reversed through the system; and this may be easily operated by a simple mechanical device with time-control. All the other advantages of the older system, of simplicity and reliability of operation, are maintained. This type of oven utilizes the principle of a heavy wall between each two oven-chambers. This wall occupies some room, and therefore more space per unit of production, yet it has several material advantages. It supports the main body of the oven-structure and carries, without difficulty, the weight of the heavy charging-cars, which, when loaded, may weigh more than 20 tons. It permits that part of the oven-structure which is subjected to the highest heats to be readily repaired, or even entirely replaced, without effecting the integrity of the main structure, and also without stopping adjacent ovens. In cases where the coal is hard on the oven bricks, or where a large percentage of water, as in washed coal, causes a severe shock to the red-hot bricks when charged, or after the plant has become old, this is an important feature. Another value of these intermediate walls is that they act as a reservoir of heat. They accumulate heat during the coking process, and when fresh coal is charged they come to the assistance of the burning gas, and help the oven pick up its heat and maintain a coking temperature.

While the earlier by-product oven-plants were simpler in design and equipment, the modern plant is almost entirely mechanically operated, and electrically-driven machinery has taken the place of most of the hand-labor. The following data illustrate the distribution and consumption of power in a plant of a capacity of 1,300 tons of coal per day.

Daily Power Consumption in Kilowatt-Hours for Various Operations.

Lighting	599
Pumps handling ammonia liquor	390
Scrubbers and pumps in by-product recovery-plant	1,283
Coal-charging and coke-pushing	192
Coal-conveyors	393
Coal-unloading	282
Coal-storage	102
Crushing and pulverizing	287
Coke-handling	686
Pumping water	1,800
	6,014

During the development of the by-product oven in the last few years much study has been given to increasing the effectiveness and economy of the apparatus for recovering the by-products. This recovery consists essentially in the cooling of the gas, which causes the condensation of much of the tar-vapor and water-vapor, which latter brings down with it part of the ammonia. After cooling, the gas is scrubbed in contact with water, sometimes by passing it through tall towers through which water is showered, sometimes by bubbling through water in a series of shallow pans, or by other methods; the object in all scrubbing processes being to bring the gas in contact as intimately as possible with wetted surfaces, to take advantage of the strong affinity of water and ammonia. The weak ammonia liquor produced in these processes is distilled by steam, producing either the ammonia liquor of commerce, or, by more refined operations, the purer grades of ammonia. If the ammonia gas from the stills is passed into sulphuric acid, ammonium sulphate, one of the most important artificial fertilizers, is produced.

Much work has been done lately on what are known as direct processes for the recovery of ammonia. In all these processes, the gas is passed directly through a bath of sulphuric acid, thereby turning the ammonia into sulphate. Several such "direct" processes have been worked out, the difference being principally in the method employed for the removal of the tar from the gas, which is then bubbled through the sulphuric acid bath; the ammonium sulphate settles out and is removed, and the gas is ready for further treatment. At this point the light oils are removed or the gas is sent direct to the point of consumption. Several methods have been advocated for the utilization of the sulphur in the gas itself as the source of sulphuric acid for making the sulphate. These processes are somewhat complicated, and will not be described here, as they have been fully discussed in the technical journals.

One of the latest applications connecting the manufacture of coke with the other arts is the "gas-oven." This name has been applied to the by-product coke-oven when adapted especially to the manufacture of illuminating-gas. The essential modification is that the gas for heating the ovens is obtained from producers, the ovens being modified to suit this gas, so that the whole of the gas produced from the distillation of the coal in the oven is available for sale as illuminating-gas. This adaptation of the retort oven is attracting considerable attention; several plants are in operation in Europe, and three or four are beginning operation or are in the course of construction in the United States. While producing practically the same gas as other systems employed in coal-gas manufacture, the gas-oven has the advantage of producing a high quality of coke. It also has the advantage in operating-cost, due to the employment of larger units, and carbonizing more coal per unit of labor. These plants are more especially adapted to the larger installations for gas-manufacture.

In discussing the use of by-product coke for metallurgical purposes, the statement is often made that this coke cannot be as good as bee-hive coke, because the quality of the coke is injured in the manufacture in order to obtain better by-products. This is not the fact, as is borne out by the considerable number of by-product oven-plants now supplying metallurgical coke for furnace- and foundry-work. In fact, in a number of localities by-product coke is sold for foundry purposes at a premium over the best bee-hive coke on the market.

One point in which by product oven-operation has improved in recent years is in reliability. In considering the gas, for example, either for gas-engine work or for illuminating purposes, reliability of supply is an absolute essential; and while formerly the supply of gas was uncertain and undependable, contracts are now made which insure a permanent and reliable supply that can be depended upon, like any other manufactured product. The sales of oven-gas for illuminating purposes now exceed 40,000,000 ft. per day.

The manufacture of by-product coke is coming to be not only an operation for the manufacture of coke and the saving of such incidental products as may be obtained, but rather an industry where coal is distilled for the purpose of producing several products, such as coke, tar, ammonia, benzol, gas, and perhaps others, each made of the best quality, and each important in maintaining the earnings of the plant. The growth of chemical manufactures in America and the greater demand for the products which supply these industries add permanency and reliability to the market, and at the same time the by-product oven furnishes a reliable supply of raw materials to important industries, which, although in some cases still in embryo here, have attained great proportions in Europe, and doubtless will grow to large importance in this country.

To the student of social economics it appears clear that the general adoption of by-product ovens will exert its influence on commercial conditions, the steady operation of these plants being a factor in maintaining the prices of coke, and hence, indirectly, prices of iron products, thus helping not a little the elimination of those violent fluctuations in iron prices, which have been a fruitful cause of the speculations and industrial depressions which so seriously affect both the employer and the employee. These plants may also have their effect on our industrial life by influencing industries to concentrate in the larger centers in order to take advantage of the raw materials which they furnish, the supply of power which may be obtained from the gas produced, and the opportunities which the plants themselves furnish for the employment of various classes of labor.

[TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS.]

The Manufacture of Coke.

BY F. E. LUCAS, SYDNEY, N. S.

(Cleveland Meeting, October, 1912.)

THIS paper is offered with considerable diffidence, since some of the statements made may not agree with the opinions of other members of the Institute. What I give is the result of some years of experience in the operation and construction of coke-ovens, and observation of plants in America and Europe.

Although my title is "The Manufacture of Coke," I have found it impossible to do justice to the subject without touching on a number of points which, while not strictly involved in the making of coke, yet are of paramount importance from an economic standpoint. There are, indeed, many points of interest, on each of which a separate paper of considerable length might be written, but the scope of my paper will permit me only to touch lightly some of them, in the hope that what I say will promote a full discussion on the whole question. Those of us who are interested in the manufacture of coke are under an obligation to the American Institute of Mining Engineers, which has opened its meetings to the discussion of this subject. I do not believe that there is any question to-day more worthy or more in need of discussion. When geologists and mining-men tell us that, at the present rate of consumption, our coal-fields will be exhausted in a few generations, we should welcome any means that will conserve these deposits as far as possible, and give us the greatest amount of heat or power from every ton mined.

I shall try to show that much can be done in this direction through the by-product coke-oven, and that it should stand second to none as an economic factor in the industrial world.

I shall here deal exclusively with the by-product oven, except for the comparative statement I have prepared, showing the saving to be effected by coking in by-product instead of beehive ovens. It should be borne in mind that this comparison is only for one particular coal, and while other coals may give, some better and some poorer results, as far as by-products and yield are concerned, yet it will serve to show the tremendous advantage that by-product ovens have over any other type.

Coke might be called coal with the volatile matter distilled off. And yet coke, as we mean it, and as it must be produced for metallurgical purposes, must be something more than this. It must conform to certain specifications in respect to physical structure and the percentage of impurities present. We have to produce a coke hard enough to carry the weight of the furnace-burden and successfully withstand the abrasion consequent upon the passage of the stock downward through the furnace or cupola· and at the same time, in order to have the best possible quality, it must not be dense. A well-developed cell-structure is essential, I believe, to the best action and lowest consumption of coke in a furnace.

The impurities in coke are principally sulphur and ash, of which the first gives the most trouble.

In America we have in the past been very fortunate in having available large quantities of coals which are naturally of a good coking quality, and require no preparation before charging, except possibly that of crushing to a certain degree of fineness. It is just as true that the exhaustion of these coals is in sight; and it will then be necessary to use other coals of possibly inferior quality. This will necessitate putting the coal through certain processes before charging, in order to maintain the standard of coke.

If the coal is high in sulphur or ash it will have to be washed to remove as much as possible of these impurities. Washing coal consists primarily in keeping the entire mass of coal so agitated in water that the impurities, by reason of their greater specific gravity, may settle to the bottom and be removed. The mechanical devices employed for doing this are many and varied. Some are good, some fairly so, and some bad; but it is not my purpose in this paper to describe or criticise the various types of apparatus. One thing I would, however, emphasize, namely, that any one type of plant does not necessarily work to the best advantage on every coal. Each coal should be carefully studied and tested by all possible means before erecting any plant to prepare it for coking. Different coals have different specific gravities, and the ratio between the specific gravity of the coal and that of the impurities also varies greatly in different coals. The manner in which the impurities are associated with the coal also varies greatly, and this in particular will largely affect the size to which the coal should be crushed before washing. We find that some coals will give the best results in washing by crushing to a small size before going to the washer and others give the best results by going to the washer in nut-size. In crushing finely before washing, it is often the case that, where the sulphur is contained in definite strata of pyrites, a large part of the pyrites is crushed to dust or flake form, which is held in suspension in the washer and goes over with the coal. When washing the coarser coal it is (more often than not) found advisable after washing to recrush to the degree of fineness which experience shows will give the best results in the coke for that particular coal.

A further preparation, with which, so far, we have not had to bother much in America, is the compression of the coal into cakes before charging. There are coals which when charged loosely into the oven either will not bond at all, or will not bond sufficiently to make a coke that will stand handling, but which, when crushed very fine and stamped into a cake that can be pushed into the oven, will make a good coke.

It is sometimes claimed that all coal should be stamped before charging, but I cannot agree with that claim, for this reason: a large open cell-structure in the coke, provided the cell-walls are strong enough to carry the burden, gives us a calorific intensity or fierceness of combustion which it is not possible to attain by the use of a dense, close-grained coke; and compressing a coal which gives us under natural conditions such a coke as we desire, only closes in the cell-structure and makes a heavier and denser coke, so that the consumption of coke in a furnace per ton of metal produced would, I believe, be higher, since practically the same bulk, and therefore greater weight, would be used.

However, this does not alter the fact that in many cases compression of the charge is of incalculable benefit to the quality of the coke produced, and as our better coking-coals become exhausted, we shall doubtless find it necessary to resort more largely to both washing and compression, in order to keep up the coke-standard. Of course, that will make the coal going to the ovens cost more than at present; but we shall find that other savings can be effected which will more than offset this added cost.

Both washing and compression are almost universally practiced in Europe, but in spite of this we find that the consumption of coke per ton of metal produced is about on a par with the general practice in America, when the quality of the ores and other conditions are considered. Also, we find that the European countries, particularly England, Germany, and Belgium, can, in spite of the fact that the bulk of their coal has to be prepared for coking, and that they are, in general, using much leaner ores, compete very successfully in the world's markets with America, which, as yet, has not been, to any great extent, confronted with the conditions which obtain on the other side. What they have done we can do, and one of the chief reasons why they can compete is because of their avoidance of waste, and their economy in small things. This is most noticeable in the general adoption of by-product ovens and the saving of all possible by-products.

The following description of the process of making coke and recovering by-products will, I think, be applicable to all types of by-product ovens:

The coking-chamber is rectangular in shape, from 30 to 40 ft. long, 6 to 12 ft. high, and 16 to 24 in. wide, taking coal-charges of from 5 to 15 tons per oven. The walls are built to contain an arrangement of flues and chambers in which the gas is burned to keep the oven at the required temperature. Underneath the oven-structure are the regenerators, through which the burnt gases pass after heating the oven-walls, the checker-work in the regenerators absorbing the heat from these gases on their way to the stack. At regular intervals the gas is reversed and passed through a second regenerator, and the air for combustion of the gas is drawn up through the one just heated. By this means air at a temperature of 1,000° to 1,800° F. is supplied for combustion of the gas.

On top of the coking-chamber is a standpipe, or ascension-pipe, connected with the gas-main leading to the condensing-plant.

The coal-charge is either dropped in through holes in the top of the oven, or, if compressed, is pushed in through the end door; and then the oven is tightly sealed, to avoid loss of gas, or the possible drawing-in of air, which would lead to a partial combustion of the charge and dilution of the gas. The gas driven off ascends through the standpipe into the collecting-main, and is then drawn to the condensing-plant by means of suitable exhausters.

The recovery of by-products in their first form takes place in the condensing-plant. During the past few years there have been so many new developments in by-product recovery-apparatus that it will be necessary to outline briefly the different processes by means of which the tar and ammonia may be extracted from the gas. The older method was to pass the gas through a series of coolers, bringing it down gradually to the vicinity of from 80° to 90° F., then pass it through the tar-scrubbers, and then through the ammonia-washers, where the gas was brought by various means into intimate contact with water. The water absorbed the ammonia, and this ammonia liquor was then distilled by steam, the ammonia vapor being driven off, and, in turn, absorbed in a bath of sulphuric acid, in which the ammonium sulphate was precipitated, and afterwards extracted and dried, ready for shipment.

The next process was a semi-direct one, in which the gas was cooled sufficiently to allow the tar to be extracted by the ordinary cooling method, and then the gas was heated up again and all passed through the sulphuric acid bath, which absorbed all the ammonia, while the gas passed on to the holder or oven. The sulphate in this case was precipitated and prepared for shipment as in the previous process.

In the semi-direct process a certain amount of weak ammoniacal liquor, consisting of the moisture which was condensed out of the gas in the original cooling, has to be dealt with in stills, the same as in the first process. The quantity of the liquor to be so dealt with will depend on the moisture in the coal charged into the ovens.

A third—the direct—process, consists in taking the gas direct from the ovens without cooling, extracting the tar at this high temperature by specially designed apparatus, and then passing the whole of the gas through the acid bath, as in the previous process. By this method there is no liquor to be dealt with and there is also no lime needed in the process, as in the case where liquor has to be distilled and a certain amount of milk of lime has to be added to release the so-called fixed compounds of ammonia in the liquor.

There is still another process, as yet in the experimental stage, in which the gas is cooled and the tar extracted as in the process first described, but the ammonia is recovered in the form of sulphate by utilizing the hydrogen sulphide in the gas instead of sulphuric acid.

A detailed description or discussion of all the different types of ovens and by-product recovery-apparatus would occupy more space than I believe could be granted for this paper.

The coke is discharged from the ovens by means of a ram, and is received either on a quenching-wharf or in one of the many types of quenching-cars or machines. In one type the entire bottom of the oven is dropped, letting the charge fall out into a quenching-car underneath.

The quenching of the coke is important from both the standpoints of moisture-content and of strength. If the coke gets too much water, the cells will suck themselves full as they cool, and there will not be heat enough left to expel this excessive moisture. At the same time, the rapid formation of steam has a tendency to crack the coke and weaken it materially. The ideal quenching of coke can, I think, be shown by immersing a piece in water, just for a moment, immediately after it comes from the ovens, and then leaving it for about 20 min. The water has just chilled the outside and stopped any tendency to combustion there. The inside will die out for want of air, and at the same time there will be sufficient heat to dry out the moisture from the outer cells. It is probably not possible in actual practice to get such results as we would in a small experiment, but the nearer we approach to it, the better the product will be.

The effect of moisture in the coke is one on which opinions may differ. Naturally, if one is buying coke by weight and has to have it hauled a considerable distance, one does not care to pay for the moisture-content or pay freight on it. Even if the moisture is determined and only dry coke paid for, there is still the question of freight, which is an important one when large quantities of coke are being used.

The effect of moisture in the coke on the action of the furnace or cupola itself, is not so clear to me. In the foundry-cupola, where the ratio of coke to iron melted is so low and the distance between the top and the melting-zone so short, I can readily see how the question of moisture may be very important; but in the blast-furnace we have a very different condition. I do not see how a reasonable amount of moisture in the coke can affect the melting-zone of the furnace or the reducing action of the gas on the ore. There is a temperature at the top of the furnace, where the coke is charged, high enough to thoroughly dry out the moisture before the coke gets far enough down to do any good, and the moisture is carried off with the gas. In fact, I believe there are some furnace-plants where the coke is sprayed with water before charging, in order to keep the top heat down below the danger limit.

Then comes the question of the effect of the moisture on the gas itself. As long as the moisture is not high enough to limit the heats required to be raised on the stoves, I cannot see that it has done any harm.

The effect of longer or shorter coking-hours or higher or lower temperatures on the quality of the coke is something that can only be determined by actual experiment with each different kind of coal. We can make laboratory-tests on a small scale to test the coking-qualities of a coal and its yield of by-products, but I would not care to build a plant to deal with a certain coal and base my calculations entirely on laboratory-tests.

In selecting a design of oven for coking, regard should be had to simplicity as far as possible, accessibility for inspection and repairs, such arrangement of combustion-chambers and flues as will give a uniform temperature in all parts of the oven; and a by-product plant that will turn the gas out free from tar, and with only traces of ammonia, at the same time producing ammonium sulphate of good color: all with due regard to the special conditions presented by a given coal and a given locality.

The advantage of a by-product oven over other types and its value as an economic factor lie not only with the coke, in that it gives a higher yield of coke from a given weight of coal, but also in the by-products and surplus gas.

The ammonium sulphate recovered finds a ready market, and the demand for this product as a fertilizer must always increase as the population of the country increases, and the farm- and garden-lands require more fertilization.

The tar also finds a ready market, and this also is bound to increase. The value of tar by-products is rising as the demand increases. It would take a book in itself to describe the many products made from tar, all the way from pitch to drugs and perfumes. But we will consider the main products alone, viz., pitch, creosote oils, and light oils. For road-making, roofing, and briquetting, there will always be a market for the pitch. With the exhaustion of our timber-lands in sight, we must either do something to conserve them or find a substitute. Railroad companies and mine-owners will find it to their advantage to creosote all railroad-ties and pit-timbers as well as bridge- and pier-timber. The lighter oils will be used in internal-combustion engines.

Probably the greatest waste the country has ever seen, or ever will see, has been going on for all the years we have been making coke in bee-hive ovens and burning the gas out in the air. Making coke in by-product ovens, and utilizing the gas by the most economical means, will revolutionize the production of power. It has already done so, most notably in Germany; and we must come to it sooner or later. The sooner we arrive at that point the better for the industries of the country, and the more we shall have done towards conserving our coal-resources.

A modern by-product oven, run at a reasonable capacity, will give 50 per cent. or more of surplus gas from a coal of about 28 per cent. volatile-content. The surplus gas is the gas over and above the quantity needed to keep the oven up to the required temperature. This surplus gas should run from 450 to 500 B.t.u. per cu. ft. The quantity of surplus gas is approximately 5,000 cu. ft.; hence, 5,000 × 450 = 2,250,000 B.t.u. per ton of coal carbonized is available for the production of power, = 93,750 B.t.u. per hour. The builders of gas-engines tell us we can get 1 h-p. on a heat-consumption of 11,000 B.t.u. On that basis, we find 8.5 h-p. per hr. from the surplus gas from 1 ton of coal.

The surplus gas can also be used for illuminating purposes. This is done at some plants in this country and at a great many in Germany. By installing two collecting-mains on top of the ovens, the rich gas, given off during the earlier hours of the coking time, can be collected in one main, and the leaner gas in the second. By this means gas of 650 to 750 B.t.u., or from 16 to 19 candle-power, can be delivered direct from the ovens without enriching. The lean gas is still of sufficiently high calorific value for heating the ovens. Gas from by-product ovens can be piped for hundreds of miles if necessary. Again, the gas may be used for steam-raising or for heating all manner of furnaces, or, in conjunction with steel-works, can be used in a steel-furnace instead of producer-gas. The recent investigations by Professor Bone have shown how by flameless combustion we can get 95 per cent. of efficiency out of the gas we burn.

In any of the above ways the gas can be used with great economy; but I believe the production of power from gas-engines opens up the largest field.

In the year 1911 there was produced in America approximately 29,338,000 tons of coke, of which approximately 21,448,000 tons was produced in bee-hive ovens. I do not know the figures for the average volatile-content of the coal that went to make this coke, but assuming a fairly low volatile coal of say 24 per cent., to produce 21,000,000 tons of coke in by-product ovens would take about 26,000,000 tons of such coal. Allowing the small amount of 4,000 cu. ft. of surplus gas per ton of coal, and 15 lb. of ammonium sulphate, and 7 gal. of tar, and allowing the surplus gas to furnish only 400 B.t.u. per cu. ft., we find that we would have 26,000,000 × 4,000 = 104,000,000,000 cu. ft. of gas per year; this at 400 B.t.u. per cu. ft. = 41,600,000,000,000 B.t.u.; reduced to hours = 4,748,858,447 B.t.u., at 11,000 B.t.u. per h-p.-hr. = 431,714 h-p. Or, allowing the value of 10 cents per 1,000 cu. ft. for the gas, we have the sum of $10,400,000. Of ammonium sulphate we have 174,107 gross tons at a value of approximately $60 per ton = $10,446,420. Of tar we have 182,000,000 gal., worth, at 2 cents per gallon, $3,640,000, Total value of gas, ammonia, and tar, $24,486,420.

I do not doubt that the coal from which the coke was made would have given better results than I have shown here, but even at these conservative figures we can see what a loss there has been.

The above amount is 7 per cent. on about $350,000,000, a sum which would build by-product ovens enough to carbonize 125,000,000 tons of coal yearly. Besides this loss, there has been the loss of the coal burned in the bee-hive oven. Allowing 64 per cent. as a fair yield for the bee-hive and 78 per cent. for the by-product ovens, there would be a loss exceeding 6,000,000 tons of coal. This at $1 per ton added to the other loss gives us a grand total of over $30,000,000 lost in one year. It seems to me that this is well worth "getting after." Much can be done if we approach this subject in the same manner, and give it the same study and attention as has been given in past years to questions pertaining to mining and metallurgy.

COMPARISON BETWEEN BEE-HIVE AND BY-PRODUCT OVENS.

Bee-Hive.

Ordinary type, 12.5 ft. in diameter.

Cost from $700 to $1,200 per oven.

Produces 4 net tons of coke in 48 hr. = 2 net tons in 24 hr.

Yield of coke from coal, 60 per cent.

By-products and surplus gas = none.

By-Product Ovens.

Oven charge, 9 tons.

Coking-time, 24 hr.

(Ovens may be larger or smaller than this, but 9 tons would probably be about the average charge for the modern type of oven.)

Coke produced on 70 per cent. yield = 6.3 tons of coke per oven in 24 hr.

By-Products.

Ammonium sulphate, 22 lb. per net ton of coal = 31 lb. per net ton of coke. Value, 2.25 cents per lb. above cost of manufacture = 70 cents per ton of coke made.

Tar. 8.5 gal. per ton of coal = 10.7 gal. per ton of coke, at 2 cents per gal. = 21 per ton of coke.

Surplus gas, 5,000 cu. ft. per ton of coal = 7,143 cu. ft. per ton of coke, at 10 cents per 1,000 cu. ft. = 71 cents per ton of coke.

Total Value of By-Products as above.

Ammonium sulphate	$.070
Tar	0.21
Gas	0.71
	$1.62 per ton of coke.

Add to the above the difference between 60 per cent. yield in bee-hive ovens and 70 per cent. in by-product ovens on the same coal. Taking coal at $1.50 per ton:

Coal per ton of coke produced in bee-hive oven	= $2.50
Coal per ton of coke produced in by-product oven	= 2.14
Balance in favor of by-product oven	= $0.36

So that the total saving in coal and by-products equals $1.62 plus $0.36 = $1.98 per ton of coke made, = $12.47 per oven in 24 hr. = $4,551.55 per oven per year. Or for by-products alone, without saving in coal, $3,723 per oven per year.

For a plant of 100 ovens, saving = $455,155 per year.

Cost of 100-oven plant complete, approximately $1,000,000.

A 100-oven plant of above capacity will produce 630 tons of coke per day = 229,950 tons per year, working on 24 hr. coking time.

If benzol is recovered it will further add to the income from by-products.

SUBJECT TO REVISION.

[TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS.]

Blowing-In a Blast-Furnace.

BY R. H. SWEETSER, COLUMBUS, OHIO.

(Cleveland Meeting, October, 1912.)

THERE are probably as many variations of the method for blowing-in blast-furnaces as there are furnace superintendents. That some of these variations are poor practice is shown by the troublesome and sometimes disastrous experiences of many furnace-men, even within the last few years. It is the object of this paper to put on record a blowing-in practice that has gradually developed to the point where the results are sure, safe, and satisfactory; and also to bring into discussion several variations that are questionable.

The blowing-in of a modern blast-furnace is one of the greatest of metallurgical operations. On its degree of success depends much of the future life of the lining and the subsequent behavior of the furnace; and it involves possibilities of danger to the men engaged in the work. The objects to be attained are the heating of the interior of the furnace to the high temperature necessary for smelting iron; the starting of the smelting process; and the gradual movement of the stock in its downward course. The most critical points are the placing of the kindling-wood, the "bringing down" of the gas, the closing of the iron-notch, and the first cast. The lighting of the first gas, the opening of the cinder-notch for the first flush, and the running of the first cast are anxious times for the superintendent and the furnace-men.

Drying-Out the Furnace.

As soon as a furnace-lining is completed it should be thoroughly dried out, whether it is to be blown-in soon or not; and the longer it is dried, the better for the furnace. At least two weeks is desirable; but good results can be had with 10 days of firing. The crudest and least desirable method of drying is to build a wood fire in the bottom of the furnace, throwing the wood in through a tuyere-arch, and using the iron-notch for draft. The accumulation of wood-ashes on the hearth-bottom keeps the brick from heating thoroughly, and considerable time is lost in removing the ashes.

An effective and easily-controlled method is to build a "Dutch oven" outside one of the tuyeres and use coke for fuel. This method may be intermittent or continuous, a desirable feature, since sometimes it is necessary to pack plates or coolers on the day-turn, and dry-out on night-turn. Coal could be used for fuel, but coke is cleaner and gives a hotter fire.

The "Dutch oven" can be built at the iron-notch, but often it is necessary to fit up the iron-notch gun and the gun-bar; and then the oven would be in the way. With the oven at a tuyere-arch, the hearth-bottom can be well heated by a wood fire built right in the casting-trough; the strong draft will draw the heat through the iron-notch. Fig. 1 shows a simple oven for a tuyere-arch; if used at the iron-notch, the neck of the flue would have to be turned downward.

Pipe for the Iron-Notch.

In order to burn completely all the kindling-wood below the tuyeres before the slag falls into it, and also in order to heat up the hearth-bottom thoroughly, it has long been customary to place an iron pipe, 3 or 4 in. in diameter and about 10 ft. long, in the iron-notch. This pipe should project from 6 to 18 in. inside the furnace-wall. If the hearth-bottom is much below the level of the iron-notch there is no need for the pipe to project much beyond the inside of the walls, but if the hearth-level is almost up to the bottom of the iron-notch, the pipe should project farther. In either case the part of the pipe inside the furnace should be thoroughly covered over with stiff clay, so as to protect it from any molten slag or iron.

The pipe should be in the center of the iron-notch, and as nearly level as possible. Near the outer end of pipe there are four holes, 0.75 in. in diameter, in which to insert hooks for pulling out the pipe when the time comes. As soon as the blast is on the furnace, the gas will come through the pipe out into the iron-trough, where there should be a wood fire to ignite it immediately. As the volume of gas increases, the gas-flame at the end of the pipe gets hotter and roars more, but does no harm. It is my practice to let this gas burn at the iron-notch for 4 hr. after starting the blast, and then to pull the pipe and shut the notch.

Shutting the iron-notch is one of the critical stages of blowing-in; but if everything is in readiness the work can be done in

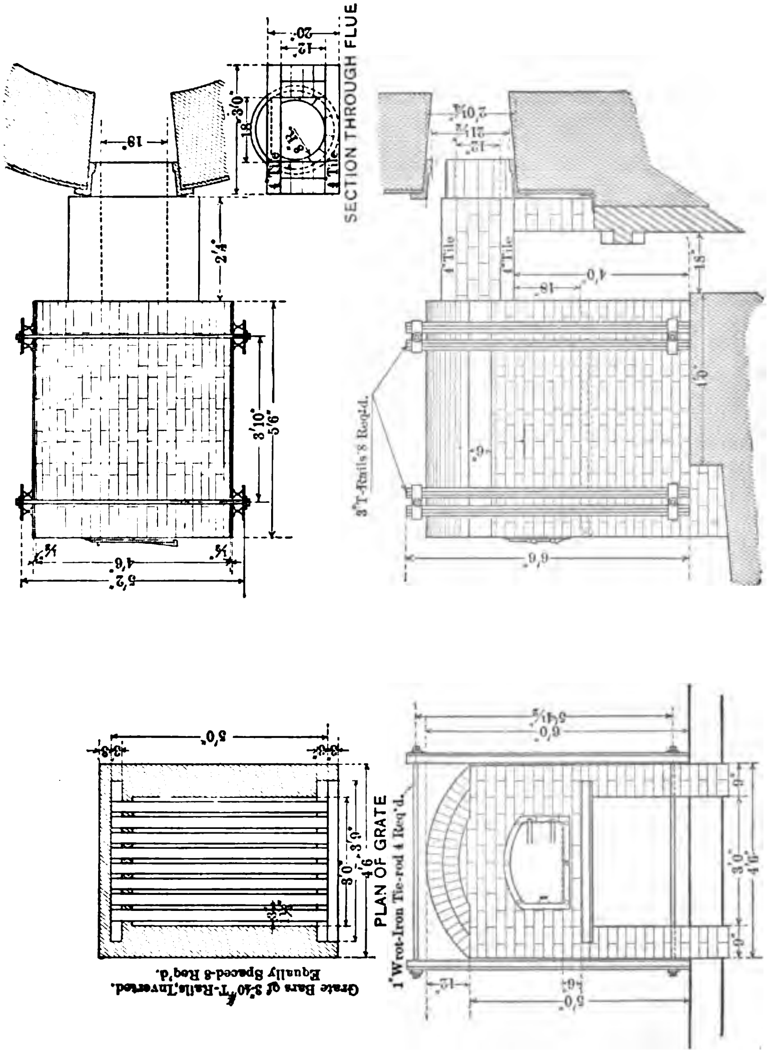

FIG. 1.—"DUTCH OVEN" FOR DRYING-OUT BLAST-FURNACES.

less than 3 min., with an iron-notch gun and a steam-cylinder attachment on the gun-bar. The iron pipe should be turned in the iron-notch so as to loosen the clay; this can be done by inserting a bar or hook in one of the four holes near the outer end of the pipe. A quick jerk on the hook will pull the pipe clear of the iron-notch, and then the gun is swung around quickly and forced into place with the steam-cylinder.

Filling the Wood.

The placing of the kindling-wood in the hearth and bosh is such an important part of blowing-in a blast-furnace, that the superintendent or his assistant usually gives personal attention to each detail, staying inside the furnace to see that every stick of wood is properly placed. The simplest method (as is often the case in furnace-practice) of putting wood in the hearth has proved to be the best in every way. David Baker taught it to me 20 years ago, and he doubtless learned it from some old furnace-man at Steelton. No fancy scaffold is made, but cord-sticks are packed as tightly as possible, in alternate layers at right angles to each other, from the hearth-bottom up to the level of the bottom side of the tuyeres. The wood is all passed into the furnace through one tuyere-arch with the cooler left out.

At the Algoma Steel Co., I first used the "Algoma cones" at the tuyeres. These are made by placing dry building-laths in front of each tuyere with the inside ends coming to a point towards the center of the furnace, and the other ends placed all around the nose of the tuyere. The spaces between cones are filled up with short sticks, split cord-sticks and any dry kindling available, and the whole surface is leveled with the exception of a place about 3.5 by 4 ft. in front of the tuyere where the wood is passed in. A vertical shaft about 10 ft. high is here made by nailing four posts (4- by 4-in. sticks or small saplings) in position. Two posts will be close to the wall on either side of the tuyere-arch and the other two will stand about 4 ft. from the wall. Cord-sticks are then piled on end all across the furnace and around the shaft; then a second tier of cord-sticks is placed in as tightly as possible. Usually the top of this second tier will be below the top of the bosh, and cord-sticks are laid against the walls all around the furnace to protect the brick and plates from falling stock.

When this is done enough wood is passed inside the furnace and up the shaft so that when the last man "backs out" he can fill the shaft solid with wood. All the men except two go outside, and the tuyere-cooler is set in place. The two men pack the cooler from inside with clay, and place the kindlings in front of the tuyere, then "back out" up the shaft. The wood is then thrown down into the shaft and packed in solid, and the two men are hoisted out at the top of the furnace in a boatswain's chair.

Filling with wood in the above manner accomplishes perfectly the three things desired, namely: the heating-up of the hearth without leaving any appreciable amount of ash; the ignition of the coke-charge and the starting of the smelting process; and the movement of the stock with such gentleness that the settling on top, within 2 hr. after the blast goes on, cannot be detected except by using the gauge-rod.

Coke on the hearth-bottom and built-up scaffolds do not usually attain all these points, and do not produce good iron on the first cast.

Filling the Stock.

On top of the wood, regular charges of coke with just enough limestone in each charge to flux the ash of the coke, are dumped in the usual manner. The size of this coke "blank" is more often too large than too small. I am indebted to Frank P. Howe, of Philadelphia, for learning the advantages of using about 20 tons of coke instead of 40 or more. The next set of charges has an equal amount of coke plus cupola-slag, and enough limestone to flux the coke and slag. Then follow the ore-charges with gradually decreasing amounts of slag. Charcoal-furnace slag is excellent for the slag-charges, but is usually harder to get than clean cupola-slag. If the slag is not granulated it should be broken up to about 2 in. size. A good volume of easily-melted slag is needed to fill up the hearth and receive the first iron that comes down.

A charging schedule that has worked successfully several times is as follows:

Charges.	Coke. Pounds.	Limestone. Pounds.	Cupola-Slag. Pounds.	Ore. Pounds.
6	6,510	730	0	0
6	6,510	4,000	4,000	0
5	6,510	4,300	4,000	1,800
5	6,510	3,600	3,000	2,400
5	6,510	3,000	2,000	3,600
5	6,510	3,300	2,000	5,000

These materials filled a 75- by 18-ft. furnace.

The first five charges after the blast went on had 1,000 lb. of slag and 6,400 lb. of ore; then the slag was omitted and the ore gradually increased to full burden.

When the stock gets within about 20 ft. of the top, it is a good plan to take measurements of the surface of the stock, and to make careful observations of the distribution; at this time, gauge-rods should be tested as to accuracy in readings.

Putting the Blast On.

I believe in putting the blast on at once, and using hot blast to light the furnace whenever stoves are available. If the volume of blast is increased properly, the furnace will not "hesitate," but will move off easily, and the first slip will be indefinitely postponed. If the stoves have not been previously heated with gas, then red-hot rods are thrust in through the tuyeres to light the kindlings.

If the furnace is a single stack and the gas-flues are cold and empty, the method of "bringing down the gas" is different from the way when there are one or more furnaces in blast and delivering gas into the same gas-mains.

When the blast is turned on a single stack, both bells and the top bleeders are left open for a while and not closed until there is a steady and voluminous flow of gas at the top of the furnace. The big bell, and then the little bell, are closed, and all the gas goes out at the bleeders. All gas-burners on stoves and boilers are tightly closed, and all fire is kept away from dust-catchers and cleaning-doors. The gas-mains are gradually filled with the dense white furnace-gas by partly closing the bleeders. At the farthest end of the gas-main, gas is allowed to escape through a valve or a cleaning-door, so as to expel all the air in the mains. When gas is leaking through all the cracks of doors and burners, and there is pressure inside the gas-main, the last gas-burner under the farthest boiler is slowly opened, and the gas is lighted by the fire on the boiler-grate. The bleeders are then closed so as to force all the gas into the mains. By having the mains full of gas at a pressure of a few inches of water, the danger of explosions is averted. The very dangerous quality of the blowing-in gas is shown by the analyses I made and recorded in our *Transactions*,[1] but all trouble can be avoided by proper handling of the gas. Great care must be taken not to open gas-burners faster than the volume of gas will allow; as the volume and pressure increase, the burners at the stoves can be opened.

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[1] In my paper, Analysis of Blast-Furnace Gas While Blowing-In, *Trans.*, xxviii., 608 to 613 (1898).

In the cases of "twin" furnaces, all these precautions can be omitted and the gas from the new furnace can be soon turned into the gas-mains by closing the bells and bleeders. Of course, the gas-valve between the new furnace and those already in blast must be opened just before the blast is turned on.

I will not attempt to give in exact cubic feet the amount of blast that should be blown during the first hours of blowing-in; but the volume at all times during the first day or two should be sufficient to prevent much blast-pressure on account of "hesitating" in the settling of stock. While the wood is burning the blast-volume should be sufficient to prevent gas from coming back in the tuyeres by reason of the pulsations of the engines. I usually start with 25 per cent. of the normal blast.

The First Flush and First Cast.

The first flush is anxiously looked for in 10 or 12 hr. after the blast goes on. Where slag is used in filling the furnace, a considerable amount should be flushed out before any attempt is made to open the iron-notch.

In opening the iron-notch for the first time there should be no trouble whatever, providing the keeper drills in on the level and through the soft clay that fills the hole left by the 4-in. pipe. Sometimes a big tapping-bar is driven into the soft clay as soon as the iron-notch is shut for the first time, as described above. But I have found that there are several disadvantages in this and now prefer to leave the bar out.

Some Records with the "Columbus Method."

Date.	July 15, 1908.	Aug. 17, 1909.	May 21, 1912.	Aug. 20, 1912.
Size of furnace, feet......	75 by 18	75 by 17	75 by 18	75 by 18
	Single.	"Twin."	Single.	"Twin."
Blast on......................	8.03 a.m.	8.25 a.m.	4.03 a.m.	3.42 p.m.
How lighted...............	Hot rods.	Hot blast.	Hot blast.	Hot blast.
Stock settled................	9.45 a.m.	9.20 a.m.	6.10 a.m.	5.20 p.m.
Iron-notch shut............	.... 12.35 p.m.	11.05 11.11 a.m.	8.00 8.05 a.m.	8.22 8.25 p.m.
First flush...................	6.45 p.m.	6.37 p.m.	4.20 p.m.	8.00 p.m.
First cast.....................	10.10 p.m.	9.25 p.m.	7.30 p.m.	12.30 p.m.
Silicon, per cent...........	4.33	2.73	3.53	1.79
Sulphur, per cent..........	0.044	0.028	0.017	0.035
Tons pig first week........	1,302	1,137	1,410	1,263

The furnace blown-in Aug. 17, 1909, had very hot stoves to start with, and the blast-temperature exceded 1,400°. On this account the blast-volume was increased more rapidly than usual, and the first cast (34,620 lb.) was only 13 hr. after the blast went on.

The furnace blown-in Aug. 20, 1912, had no bottom-brick on top of the salamander that had been left in, and on account of the increased depth of hearth the cinder was longer than usual in filling up to the cinder-notch.

[SECRETARY'S NOTE.—For earlier contributions to the Institute covering this subject in whole or in part, see the following papers in the *Transactions*.—J. S.]

Vol.	Page.	Title.	Authors.
VIII.	348 to 354	Blast-Furnace Working.	Julian Kennedy.
XVII.	138 to 140	Large Furnaces on Alabama Material.	F. W. Gordon.
XVIII.	379 to 391	Filling and Blowing-In at the Durham Blast-Furnace.	B. F. Fackenthal, Jr.
XVIII.	427 to 436	Peculiar Working of a Blast-Furnace.	N. B. Wittman.
XX.	262 to 267	American Blast-Furnace Practice (Discussion).	James Gayley, J. E. Johnson, Hugh Kennedy, John Birkinbine, E. C. Pechin, Julian Kennedy.
XXIII.	370	Review of American Blast-Furnace Practice.	E. C. Potter.
XXVIII.	608 to 613	Analysis of Blast-Furnace Gas While Blowing-In.	Ralph H. Sweetser.

SUBJECT TO REVISION.

[TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS.]

Note on the Case-Hardening of Special Steels.

BY ALBERT SAUVEUR* AND G. A. REINHARDT,† CAMBRIDGE, MASS.

(Cleveland Meeting, October, 1912.)

ALTHOUGH many metallurgists know that some pearlitic special steels can be made troostitic, martensitic, and even austenitic, without quenching, and, therefore, without exposing them to the dangers of the quenching-bath, the practical significance of such possibility does not seem to have received the attention it deserves. Let us recall briefly the mechanism of the transformation alluded to.

In Fig. 1 (after Guillet) is shown the constitutional diagram of nickel-steel that; is, of alloys of iron, carbon, and nickel. Any steel the composition of which falls within the area *LPO* is pearlitic after slow cooling; any steel the composition of which is represented by a point within the area *MLO* is martensitic, and any steel within the area *MNO* is austenitic. Starting with pearlitic steel, then by increasing the carbon-content or the nickel-content, or both, the metal becomes martensitic and eventually austenitic; that is, its composition crosses first the boundary-line *LO* and then the line *MO*—and it remains martensitic or austenitic after slow (air) cooling.[1] We have in this way imparted to the metal without quenching it the properties of hardened carbon-steel. In other words, the steel is now self-hardening. Nickel cannot be introduced into solid steel, but by the well-known process of case-hardening it should be possible to cause pearlitic nickel-steel to absorb the necessary amount of carbon to become martensitic, or even, if desired, austenitic. It should therefore be possible to manufacture case-hardened objects of nickel-steel and of some other special steels having pearlitic, and consequently soft and tough, cores, and troosto-martensitic, martensitic, martenso-austenitic or austenitic cases, without recourse to the quenching-bath. In Fig. 1, for instance, the point *A* represents a steel containing 0.25 per cent. of carbon and 3 per cent. of nickel. By increasing the carbon-content to 1.2 per cent. its composition shifts from *A* to *B* and the metal becomes martensitic. Increasing its carbon to 1.5 per cent. would make it austenitic, since its composition would now be represented by the point *C*. A pearlitic steel containing the same amount of carbon (0.25 per cent., but more nickel, say 5 per cent., would require a smaller addition of carbon to become martensitic or austenitic, since by shifting its composition from *A′* to *B′* when it contains 0.85 per cent. of carbon it fails within the martensitic area. It is evident that the nearer the composition of pearlitic steel to the boundary-line *LO* the less carbon will it require to make it martensitic.

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* Professor of Metallurgy and Metallography in Harvard University.

† Assistant in Metallurgy and Metallography in Harvard University.

[1] The boundaries between the various zones are not, as a matter of fact, as sharp as indicated in the diagram, because pearlite is not transformed abruptly and in its entirety into martensite when a certain critical composition is attained, nor is martensite suddenly and completely converted into austenite. Greater refinement in the construction of the diagram would undoubtedly result in the introduction of troosto-pearlitic, troostitic, and troosto-martensitic zones between the pearlitic and martensitic areas and of a martenso-austenitic zone between the martensitic and austenitic areas. For the purpose of this note, however, the existence of these transition-conditions may be ignored.

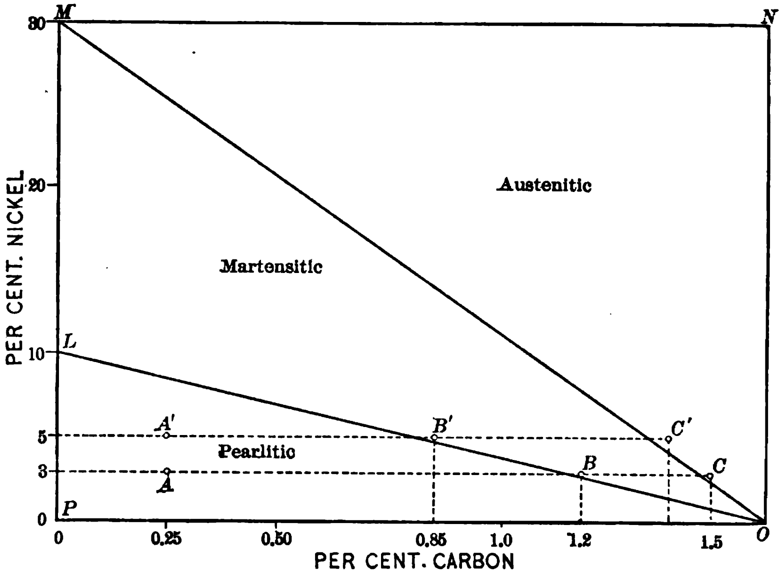

FIG. 1.—CONSTITUTIONAL DIAGRAM OF NICKEL-STEEL (GUILLET).

To test the practicality of the manufacture of case-hardened articles of the description referred to, a number of nickel-steels of varying compositions were subjected to the case-hardening treatment, and no difficulty was experienced in producing martensitic cases and even, in the presence of sufficient nickel, austenitic cases.

It will suffice for the purpose of this note to describe briefly a few of the tests performed.

Steel No. 3.

Origin.—Nickel-steel supplied by the Peerless Motor Car Co., of Cleveland, Ohio.

Shape.—1-in. hexagonal bar.

Chemical Composition.—Ni, 3.44; C, 0.176; P, 0.011; S, 0.026; Mn, 0.48 per cent.

Treatment A.—Annealed for 3 hr. at a temperature of 1,000° C., surrounded by a mixture composed of 40 per cent. of barium carbonate and 60 per cent. of wood-charcoal, and cooled in air.

Results.—Approximate thickness of case, 1.3 mm. Aproximate thickness of martensitic portion of case, 0.5 mm. The balance of the case was troosto-martensitic, troostitic, and troosto-pearlitic.

The microstructure of the case is shown in Figs. 2 and 3, magnified respectively 50 and 100 diameters. The purely martensitic character of the shell of the case will be noted, as well as the occurrence of troostite and its gradual increase. In Fig. 2 the wide black band is pure troostite. The martensitic grains owe their polyhedral form to the original austenitic grains from which they are derived. Some of the grains are surrounded by troostitic membranes.

Treatment B.—Same as treatment A, but lasting 5 hr.

Results.—Case of the same character, but measuring 1.7 mm. in thickness and the martensitic portion 0.9 mm.

Treatment C.—Same as treatment B, but the steel was allowed to cool slowly in the case-hardening furnace after the operation.

Results.—Approximate thickness of case, 1.9 mm. The case consisted of sorbitic grains surrounded by free cementite and containing many cementite needles, as shown in Fig. 4.

By referring to Fig. 1 it will be seen that in the presence of 3.5 per cent. of nickel some 1.2 per cent. of carbon is needed in order that the steel may be martensitic after air-cooling. It will be evident from the structure of the slowly-cooled sample (Fig. 4) that at least that amount of carbon had been introduced.

Conclusions.—Steel containing some 3.5 per cent. of nickel can be made martensitic by case-hardening followed by air-cooling. Slow cooling in the furnace produces a sorbito-cementitic case.

Other Treatments.—The same steel was case-hardened, using crushed bone, but the results were much less satisfactory.

Steel No. 5.

Origin.—Nickel-steel supplied by the Peerless Motor Car Co. of Cleveland, Ohio.

Shape.—1.5-in. round bars.

Chemical Composition.—Ni, 4.86; C, 0.115; P, 0.010; S, 0.037; Mn, 0.50 per cent.

Treatment A.—Annealed for 5 hr. at a temperature of 1,000° C., surrounded by a mixture composed of 40 per cent. of barium carbonate and 60 per cent. of wood-charcoal, and cooled in air.

Results.—Approximate thickness of case, 2.5 mm. Approximate thickness of austenitic portion of case, 0.5 mm. Approximate thickness of martensitic portion, 1.3 mm. The balance of the case was troosto-martensitic, troostitic, and troosto-pearlitic.

The microstructure of the case is shown in Figs. 5 and 6, magnified 100 and 300 diameters respectively. The white constituent is austenite, the dark needles martensite.

Treatment B.—Same as treatment A with the exception that the bar was allowed to cool slowly with the furnace after the case-hardening operation.

Results.—A case about 2 mm. thick was produced consisting of large grains of sorbite and much free cementite.

By referring to Fig. 1 it will be seen that in the presence of 5 per cent. of nickel or thereabout some 0.9 per cent. of carbon should theoretically suffice to produce a martensitic case, while 1.30 per cent. of carbon or more should make it austenitic. The results obtained in these case-hardening tests agree satisfactorily with the diagram.

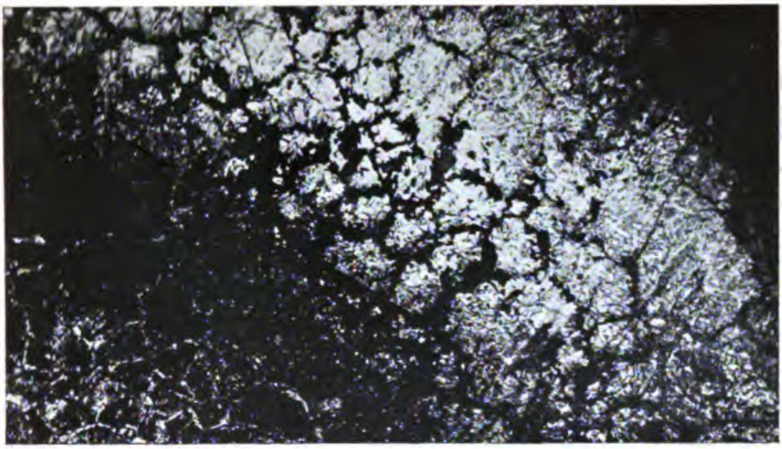

FIG. 2.—NICKEL-STEEL. NI, 3.44; C, 0.176 PER CENT. CASE-HARDENED STRUCTURE OF CASE. MAGNIFIED 50 DIAMETERS.

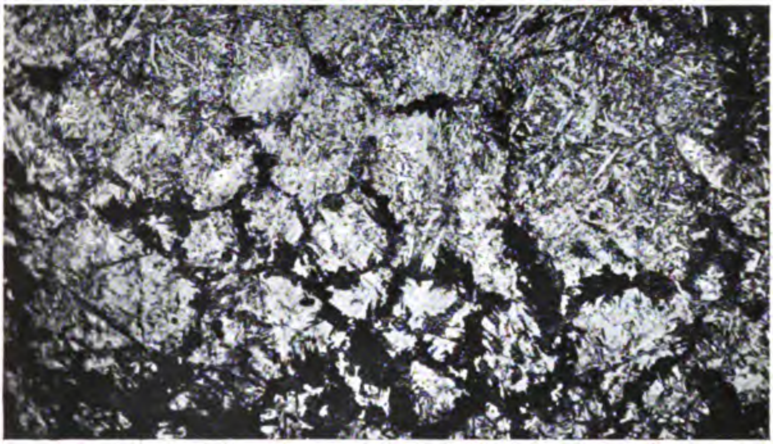

FIG. 3.—SAME STEEL AS IN FIG. 2. SAME TREATMENT. MAGNIFIED 100 DIAMETERS.

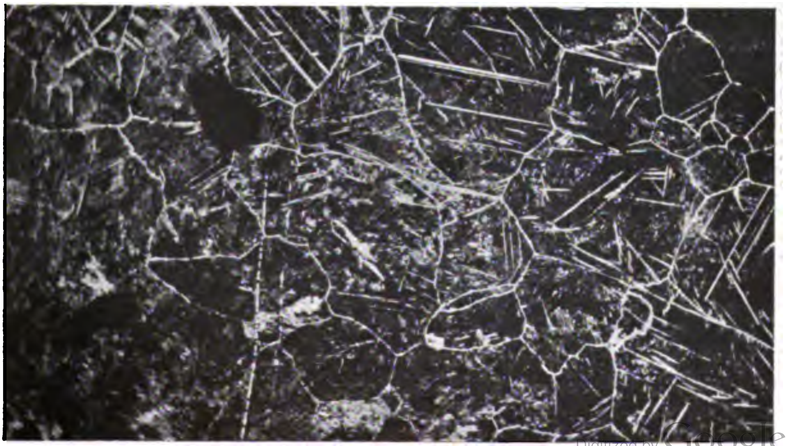

FIG. 4.—SAME STEEL AS IN FIGS. 2 AND 3, BUT FURNACE-COOLED AFTER CASE-HARDENING. MAGNIFIED 100 DIAMETERS.

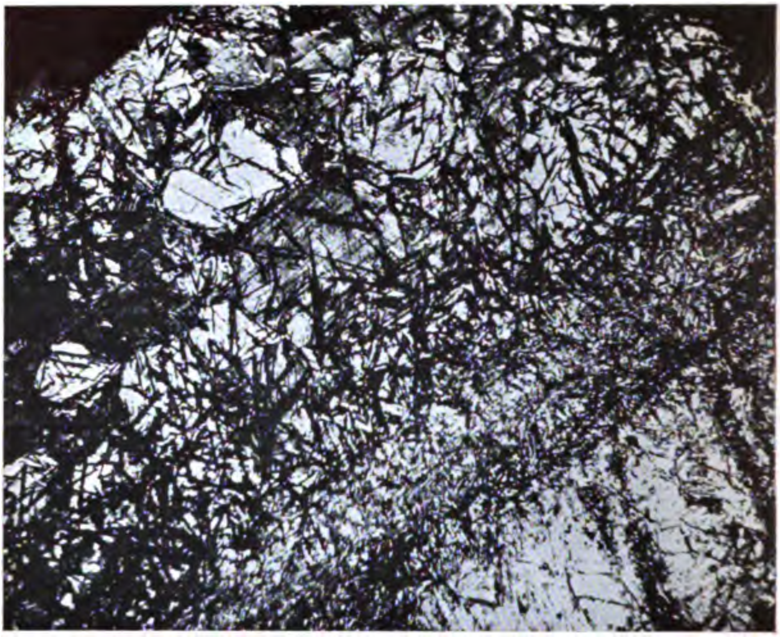

FIG. 5.—NICKEL-STEEL. NI, 4.86 ; C, 0.115 PER CENT. CASE-HARDENED. STRUCTURE OF CASE. MAGNIFIED 100 DIAMETERS.

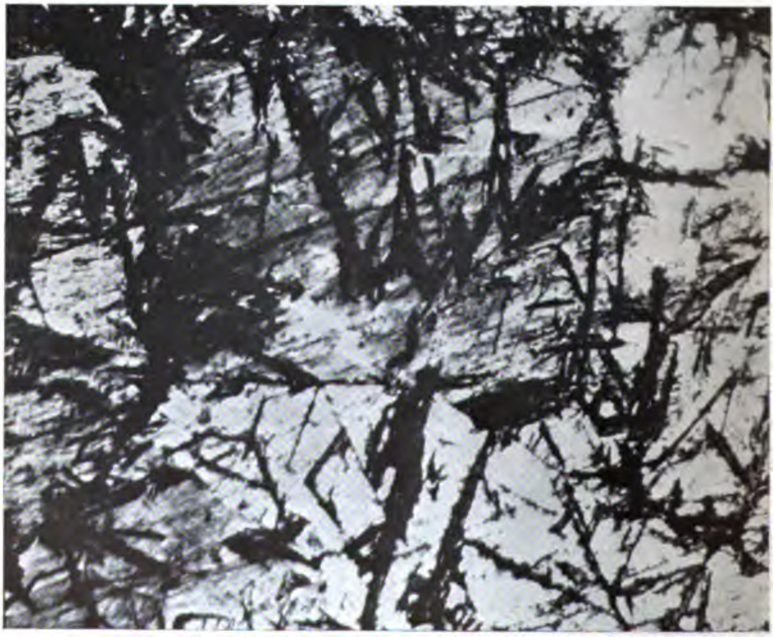

FIG. 6.—SAME STEEL AS IN FIG. 5. SAME TREATMENT. MAGNIFIED 300 DIAMETERS.

Conclusions.—In case-hardening nickel-steel containing about 5 per cent. of nickel, austenitic or martensitic cases can readily be produced through proper regulation of the length of the operation and by cooling in air the case-hardened articles. Slow cooling in the furnace produces sorbito-cementitic cases.

Treatment of Sorbito-Cementitic Cases.—By reheating to 1,000° C. a steel with a sorbito-cementitic case (because of slow cooling in the furnace) and cooling in air, the case became martensitic. This suggests the advisability of cooling in the furnace after case-hardening producing a sorbito-cementitic case and then reheating to 800° C. and cooling in air, thereby securing a finer martensitic case than by cooling in air from the case-hardening temperature.

COMMERCIAL APPLICATIONS.

The possible commercial applications of a case-hardening operation yielding martensitic cases without recourse to the quenching-bath are obvious. It should be possible to case-harden finished pieces of machinery, gears for instance, of low-carbon nickel-steel, producing hard wearing-surfaces which should be free from the cracks, distortions, and internal strains so frequently present in quenched steel.

We are indebted to the Peerless Motor Car Co. for supplying the steel used in these tests, and to R. R. Abbott, metallurgist of the company, for his interest in the work and valuable suggestions.

[TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS.]

The Alundum Extraction-Thimble Used in the Determination of Copper.

BY L. W. BAHNEY, NEW HAVEN, CONN.

(Cleveland Meeting, October, 1912.)

THE photograph, Fig. 1, shows the apparatus a little less than half size, consisting of a filtering-flask fitted with rubber stopper through which passes a bent glass tube, and an extraction-thimble fitted with rubber stopper through which passes a glass tube of 0.25-inch bore. Both tubes are connected by a short piece of rubber tubing.

A section of a thimble is shown in the photograph; the tube extends to within ⅛ in. of the tapered end.

The object of using the thimble is to remove the acid from the beaker after all the copper has been precipitated. Time is saved, the copper is not exposed to the acid alone, and there are none of the losses attending ordinary filtration. I have accomplished these results by means of a piece of perforated platinum fastened in the end of a 0.25-in. bore glass tube and a filter-mat of asbestos, but after my supply of proper length fiber became exhausted I could not replenish it even after purchasing 14 lots from four different dealers.

The above apparatus may be used to remove at least seven-eighths of a supernatant liquid from a settled precipitate without disturbing the latter.

The application of the thimble is best shown by partly outlining the assay for copper, as follows:

Dilute the acid solution of copper and other sulphates to 150 cc. in a 200-cc. Jena beaker, place on a hot plate, add 2 drops of concentrated HCl, then place a strip of aluminum in the beaker (this may be bent or straight, as desired). Connect the apparatus, as shown in Fig. 1, with a filter-pump having a strong suction. When the copper is precipitated, remove the beaker from the hot plate and insert the extraction-thimble alongside the strip of aluminum. The acid solution will be drawn through the porous tube. Wash the upper end of the aluminum strip with a jet of hot water, wash down the sides of the beaker, and add about 25 cc. of hot water to cover the precipitate.

As soon as all the liquid is out of the beaker, disconnect the thimble by slipping the rubber tubing from the bent glass tube of the filtering-flask, wash it with a jet of hot water. If there is no adhering copper, lift from the beaker, add 5 cc. of strong nitric acid to the precipitated copper, and carry out the titration in the usual manner. If the copper is in a finely-divided condition as a precipitate on the aluminum, it will adhere as a black coating on the thimble, but as soon as the thimble is disconnected the water inside it will ooze out through the pores, and if the 5 cc. of strong nitric acid be poured on it, the copper will immediately dissolve and the thimble may be washed with a jet of hot water. The filtration is more rapid when the solution is hot.

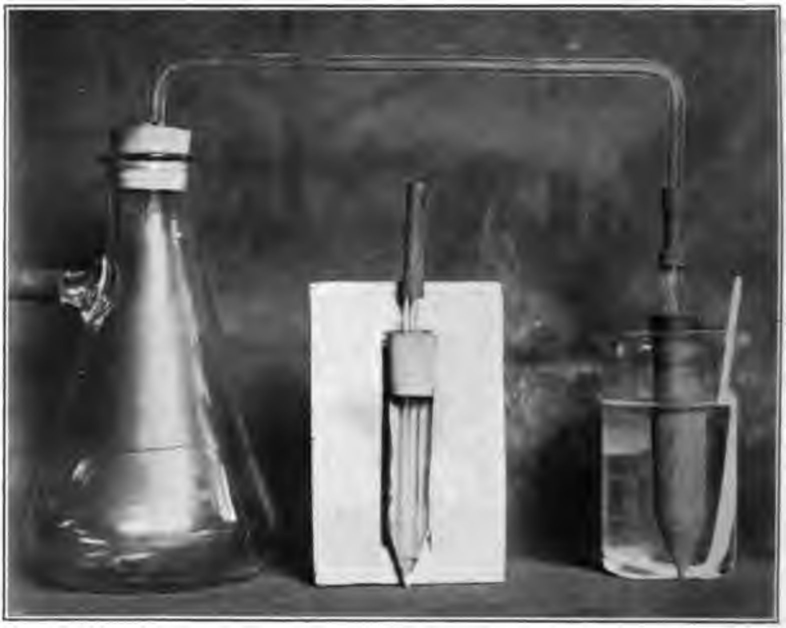

Fig. 1.—The Alundum Extraction-Thimble as Used in the Determination of Copper.

If the last of the solution in the beaker is not withdrawn as rapidly as the first, tip the beaker at an angle of 45°. With a pump having a strong suction this will not be necessary.

The thimble attached to the flask, as shown above, was used in more than 300 determinations.

[TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS.]

Proceedings of the One Hundred and Third Meeting, Cleveland, Ohio, October, 1912.

LOCAL COMMITTEE.

EXECUTIVE.—D. T. Croxton, *Chairman;* C. B. Murray, *Secretary;* F. B. Richards, W. G. Mather, A. W. Smith, S. T. Wellman.

FINANCE.—W. G. Mather, *Chairman;* H. H. Brown, C. A. Grasselli, Samuel Mather, I. B. Miller.

ENTERTAINMENT.—F. B. Richards, *Chairman;* H. W. Lash, B. D. Quarrie, W. Rattle, Jr., Willard Sawyer, R. B. Sheridan.

PROGRAM AND PRINTING.—C. B. Murray, *Chairman;* L. E. Dunham, F. A. Emmerton, G. F. Knapp, A. W. Smith, O. Textor.

GENERAL RECEPTION.—R. R. Abbott, M. A. Ammon, H. A. Barren, G. Bartol, H. H. Brown, W. N. Crafts, J. H. Cremer, B. Crowell, D. T. Croxton, S. W. Croxton, H. G. Dalton, L. E. Dunham, F. A. Emmerton, C. H. Fulton, C. A. Grasselli, H. C. Hale, L. E. Holden, R. F. Jopling, H. L. Kaufman, H. R. Kimmell, G. F. Knapp, W. Koehler, H. M. Lane, H. W. Lash, I. P. Lihme, A. G. McKee, M. McMurray, Samuel Mather, W. G. Mather, E. P. Merrill, H. G. Merry, L. B. Miller, C. B. Murray, E. W. Oglebay, F. J. Peck, T. E. Pierce, B. D. Quarrie, W. J. Rattle, F. B. Richards, L. A. Roby, H. E. Seidl, J. H. Sheadle, A. W. Smith, O. Textor, F. H. Treat, F. R. Van Horn, O. M. Weichsel, S. T. Wellman, R. Zeising.

IRON AND STEEL DIVISION.

Charles Kirchhoff, *Chairman;* Charles F. Rand, *Vice-Chairman;* Bradley Stoughton, *Secretary;* John Birkinbine, James Gayley, Henry D. Hibbard, Henry M. Howe, Charles K. Leith, Robert W. Hunt, Julian Kennedy, Richard Moldenke, Joseph W. Richards, Albert Sauveur, F. W. C. Schniewind, A. A. Stevenson, Felix A. Vogel, Leonard Waldo, William R. Webster.

INSTITUTE HEADQUARTERS, Hotel Statler.

On Monday evening, Oct. 28, 1912, the visiting members and guests were informally received by the Local Committee at the Headquarters of the Institute at the Hotel Statler. The evening was very pleasantly spent in renewing old acquaintances and friendships and in forming new ones.

The first session, held Tuesday morning, Oct. 29, 1912, in the impressive Assembly-Room of the hotel, was called to order by David T. Croxton, Chairman of the Local Committee. Mr. Croxton introduced the Hon. Newton D. Baker, Mayor of Cleveland, who extended a cordial welcome to the members and guests present. James F. Kemp, President of the Institute, who subsequently presided at the meeting, responded in behalf of the Institute.

The following papers were presented in oral abstract by the authors:

* Recent Developments in the Inspection of Steel Rails, by Robert W. Hunt, Chicago, Ill.

Discussion by Henry D. Hibbard, Frederick Laist, Elwood Haynes, William H. Blauvelt, R. H. Sweetser, and reply by Mr. Hunt.

* Notes on Titanium and on the Cleansing Effect of Titanium on Cast-Iron, by Bradley Stoughton, New York, N. Y.

Discussion by J. E. Johnson, Jr., James F. Kemp, and reply by Mr. Stoughton.

† The Cuyuna Iron-Ore Range, by Walter A. Barrows, Jr., Duluth, Minn., and Carl Zapffe, Brainerd, Minn.

Discussion by D. T. Croxton.

* The Action of Various Commercial Carbonizing-Materials, by Robert R. Abbott, Cleveland, Ohio.

Discussion by Henry D. Hibbard.

* Measurements and Relations of Hardness and Depth of Carbonization in Case-Hardened Steel, by Mark A. Ammon, Cleveland, Ohio.

Discussion by J. E. Johnson, Jr., Bradley Stoughton, and reply by Mr. Ammon.

(A unanimous vote of thanks was passed to the Peerless Motor Car Co., for their liberal policy in placing at the disposal of the Institute the information presented by Messrs. Abbott and Ammon in their excellent papers.)

* Notes on Ruff's Carbon-Iron Equilibrium Diagram, by Henry M. Howe, New York, N. Y. (Presented by Bradley Stoughton.)

‡ N. M. Wittorff's Preliminary Investigation of Primary Crystallization, and Subsequent Physico-Chemical Permutations in Iron-Carbon Alloys Containing More than 4 Per Cent. of Carbon, by Bradley Stoughton and Simon Marcus, New York, N. Y. (Presented by Mr. Stoughton.)

Discussion by J. E. Johnson, Jr., Henry D. Hibbard, Bradley Stoughton, R. H. Sweetser, James F. Kemp, and reply by Mr. Stoughton.

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* Papers distributed in pamphlet form at the meeting.

† Papers distributed in mimeograph form at the meeting.

‡ Papers in manuscript form available for consultation.

The second session, on Tuesday afternoon, held in the same place, was called to order by President Kemp.

The following papers were presented in oral abstract by the authors:

* Blowing-In a Blast-Furnace, by Ralph H. Sweetser, Columbus, Ohio.

Discussion by J. E. Johnson, Jr.

* Alloys of Cobalt with Chromium and Other Metals, by Elwood Haynes, Kokomo, Ind.

Discussion by P. N. Moore, William Kelly, Carl Zapffe, R. R. Abbott, S. T. Wellman, J. E. Johnson, Jr., James F. Kemp, Henry D. Hibbard, and reply by Mr. Haynes.

* The Manufacture of Coke, by William H. Blauvelt, Syracuse, N. Y.

* By-Product Coke, by Charles W. Andrews, Duluth, Minn. (Presented by Dr. Schniewind.)

* The Manufacture of Coke, by F. E. Lucas, Sydney, N. S. (Presented by William H. Blauvelt.)

The above three papers on the manufacture of coke were discussed by James E. Thropp, Jr., James F. Kemp, E. V. d'Invilliers, Dr. F. Schniewind, Dr. R. Moldenke (communication presented by Mr. Stoughton), Bradley Stoughton, H. H. Stoek, Charles H. Fulton, Ralph H. Sweetser, and replies by Mr. Blauvelt and Dr. Schniewind.

* The Concentration of Iron-Ores, by H. V. Hansell, New York, N. Y.

Discussion by E. G. Spilsbury, P. N. Moore, J. E. Johnson, Jr., E. V. d'Invilliers.

* The Effect of Alumina in Blast-Furnace Slags, by J. E. Johnson, Jr., Ashland, Wis.

Discussion by Ralph H. Sweetser, Charles H. Fulton, James F. Kemp, and reply by Mr. Johnson.

The third session, Wednesday morning, Oct. 30, held in the same place, was devoted to a discussion of the business affairs of the Institute. President Kemp called the meeting to order and prefaced the proceedings with a brief history of the Institute management to date, calling attention to the fact that the Institute now being an incorporated body of the State of New

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* Papers distributed in pamphlet form at the meeting.

York, by its charter can take no legal action at meetings held outside of the State of New York. The present session therefore could only be devoted to discussion.

Charles F. Rand, in behalf of the Special Committee appointed by the Board of Directors and the Council (Joseph W. Richards, *Chairman;* Charles Kirchhoff, and Charles F. Rand), presented printed copies of the proposed new Constitution and By-Laws, which were distributed among the members. The various articles were taken up one by one, special attention being given to Art. IV., which relates to the establishment of a class of "Fellows" in the Institute. This Article was submitted more for discussion than as a recommendation. Discussion from the floor was held by C. R. Corning, Charles Kirchhoff, J. W. Malcolmson, P. N. Moore, H. H. Stoek, Robert W. Hunt, J. E. Johnson, Jr., H. D. Hibbard, E. V. d'Invilliers, H. F. Bain, E. W. Parker, Bradley Stoughton, A. R. Ledoux, William Kelly, E. C. Spilsbury, Carl Zapffe, S. L. Goodale, E. H. Benjamin, Joseph Struthers.

The Article as drawn granted power to the body of Fellows to express an opinion as such with regard to "pending or proposed legislation affecting the mining or metallurgical industries." A vote was called to ascertain the sense of those present concerning this question as put. The result was 37 against and 17 in favor. The broad question of establishing a class of Fellows was then presented and voted upon, with the result of 27 against and 20 in favor.

The proposed By-Laws were briefly presented and a special discussion was held on By-Law III., referring to the duties of the Secretary; especially the section which required that the Secretary should devote his entire time to the affairs of the Institute. Discussion from the floor was held by J. E. Johnson, Jr., Charles F. Rand, A. R. Ledoux, Charles Kirchhoff, E. G. Spilsbury, S. L. Goodale, James F. Kemp, and S. W. Malcolmson. On request, an expression of opinion was asked as to whether the restriction should remain, namely, that the entire time of the Secretary should be devoted to Institute affairs; the vote showed 13 in favor of the restriction and 30 against it.

C. R. Corning presented arguments in support of the proposed amendments to the Constitution and By-Laws presented by himself and George C. Stone at the adjourned Annual Business Meeting of the Institute, Oct. 7, 1912, and subsequently sent in printed form to the membership at large in circular No. 8, Oct. 15, 1912.

The fourth and concluding session was held Thursday morning at the Case School of Applied Science in the large lecture-room of the Electricity Building. Professor Kemp presided. The following papers were presented in oral abstract:

† The Quality of Charcoal-Iron, by J. E. Johnson, Jr., Ashland, Wis.

Discussion by Bradley Stoughton and by letter from Henry M. Howe and Richard Moldenke.

* The Constitution and Melting-Points of a Series of Copper-Slags, by Charles H. Fulton, Cleveland, Ohio.

Discussion by Frederick Laist and Henry D. Hibbard.

* Chemistry of the Reduction Processes in Use at Anaconda, Mont., by Frederick Laist, Anaconda, Mont.

* Note on The Case-Hardening of Special Steels, by Albert Sauveur and G. A. Reinhardt, Cambridge, Mass. (Presented by Mr. Reinhardt.)

Discussion presented in writing by R. R. Abbott.

A discussion of the paper of R. W. Raymond on Our National Resources and Our Federal Government, by George Otis Smith, Washington, D. C. (Presented by E. W. Parker.)

C. R. Corning presented orally a few additional remarks concerning the proposed amendments to the Constitution and By-Laws, discussed by him at the third session.

The proceedings concluded with a cordial vote of thanks to the Local Committee for the complete and delightful manner in which the comfort and enjoyment of the members had been looked after; also to the Case School of Applied Science for placing at disposal of the Institute for the final session the lecture-room in the Electricity Building.

In addition to the papers already noted, the following were read by title for publication:

* Surveying and Sampling Diamond-Drill Holes, by E. E. White, Ishpeming, Mich.

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* Papers distributed in pamphlet form at the meeting.

† Papers distributed in mimeograph form at the meeting.

† The Utility of Efficiency-Records in the Manufacture of Iron, by John J. Porter, Staunton, Va.

* Fuel-Efficiency of the Cupola-Furnace, by John J. Porter, Staunton, Va.

* The Microstructure of Iron and Steel, by William Campbell, New York, N. Y.

* Recent Developments in Open-Hearth Steel-Practice, by N. E. Maccallum, Phœnixville, Pa.

† Influence of Divorcing Annealing on the Mechanical Properties of Low-Carbon Steel, by Henry M. Howe and Arthur G. Levy, New York, N. Y.

† Methods of Preparing Basic Open-Hearth Steel for Castings, by H. F. Miller, Jr., Verona, Pa.

† Electric Heating and Removal of Phosphorus from Iron, by A. E. Greene, Chicago, Ill.

* A Titaniferous Iron-Ore Deposit in Boulder County, Colo., by E. P. Jennings, Salt Lake City, Utah.

† The Function of Slag in Electric Steel-Refining, by R. Amberg, Pittsburg, Pa.

* The Ta-yeh Iron-Ore Deposits, Hu-pei Province, China, by C. M. Weld, Low Moor, Va.

* Clinton Iron-Ore Deposits in Kentucky and Tennessee, by S. Whinery, New York, N. Y.

* Melting Iron in the Cupola-Furnace, by R. Moldenke, Watchung, N. J.

* Centrifugal Machines as Applied to Ore-Grading and Ore-Concentrating, by Godfrey T. Vivian, Cornwall, England.

* The St. Helens Mining-District, Skamania, Wash., by Horace V. Winchell, Minneapolis, Minn.

* Fires in Metalliferous Mines, by George J. Young, Reno, Nev.

‡ Fire-Clay Deposits in Canada, by Heinrich Ries, Ithaca, N.Y.

‡ Discussion of F. G. Cottrell's paper, Electric Fume-Precipitation, by Richard Peters, Jr., Barker, Pa.

* Present Conditions of Mining in the District of Vladivostok, Siberia, by Albert F. J. Bordeaux, Thonon-les-Bains, France.

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* Papers distributed in pamphlet form at the meeting.

† Papers distributed in mimeograph form at the meeting.

‡ Papers in manuscript form available for consultation.

* The Sampling of Gold-Bullion, by Frederick P. Dewey, Washington, D. C.

* An Experience in the Use of Water-Power, by C. M. Myrick, San Francisco, Cal.

* Our National Resources and Our Federal Government, by R. W. Raymond, New York, N. Y.

* The Wood Flotation Process, by Henry E. Wood, Denver, Colo.

* The Occurrence of Gold in the Eocene Deposits of Texas, by E. T. Dumble, Houston, Texas.

* A Graphic Solution of D'Arcy's Formula for the Transmission of Compressed Air in Pipes, by Nathaniel Herz, Lead, S. D.

* The Alundum Extraction-Thimble Used in the Determination of Copper, by L. W. Bahney, New Haven, Conn.

* Notes on the Metallography of Alloys, by William Campbell, New York, N. Y.

‡ The Ultimate Sources of Ores, by Blamey Stevens, Temascaltapec, Mexico.

* Notes on Bag-Filtration Plants, by Anton Eilers, New York, N. Y.

* The Development of the Parkes Process in the United States, by Ernst F. Eurich, New York, N. Y.

* The Sulphatizing-Roasting of Copper-Ores and Concentrates, by Utley Wedge, Ardmore, Pa.

‡ Development of the American Water-Jacket Lead Blast-Furnace, by R. C. Canby, Wallingford, Conn.

‡ Heat-Losses in Furnaces, by F. A. J. Fitzgerald, Niagara Falls, N. Y.

* The Development of the Reverberatory Furnace for Smelting Copper-Ores, by E. P. Mathewson, Anaconda, Mont.

‡ A Trip Through Northern Korea, by H. W. Turner, San Francisco, Cal.

‡ Discussion of Mr. Keller's paper, The Solubility in Nitric Acid of Gold Contained in Certain Copper-Alloys (Copper-Bullions), by Charles Hunter, Swaziland, South Africa.

‡ Discussion of Messrs. Sauveur and Reinhardt's paper, Note

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* Papers distributed in pamphlet form at the meeting.

‡ Papers in manuscript form available for consultation.

on the Case-Hardening of Special Steels, by Robert R. Abbott, Cleveland, Ohio.

‡ New Type of Blast-Furnace Construction, by J. E. Johnson, Jr., Ashland, Wis.

‡ New Design of Open-Hearth Furnace Using Producer-Gas, by H. F. Miller, Jr., Verona, Pa.

Excursions and Entertainments.

On Tuesday evening, Oct. 29, an informal smoker in the large Assembly-Room of the Statler Hotel was tendered to the members and guests of the Institute party, and through the courtesy of Rogers, Brown & Co., a remarkably vivid motion-picture portrayal was presented entitled "From Mine to Molder," beginning with the mining of the ore and showing the methods and complete processes until the finished material is produced. A running explanation of the various steps of the process was given by Henry B. B. Yergason, a representative of the company. The smoker concluded with an interesting illustrated address by Prof. James F. Kemp and an informal collation.

On Tuesday afternoon, Oct. 29, during the technical session, the visiting ladies were given an automobile ride through the delightful boulevards of the city and later entertained with a tea at the Country Club.

On Wednesday afternoon, Oct. 30, the party was taken by a special train, provided through the kindness of the American Steel & Wire Co., to the Central and Newburg plants of that company. Mr. B. D. Quarrie, the General Superintendent, and a corps of assistants piloted the party in groups through both plants. The return to headquarters was made by special trolley-cars.

On Wednesday evening a banquet to more than a hundred was held in the Assembly-Room of the Hotel, which was tastefully decorated with flowers. Mr. D. T. Croxton, the genial toast-master, contributed largely to the enjoyment of the occasion. Informal addresses were made by Capt. Robert W. Hunt, W. G. Mather, E. W. Parker, E. H. Benjamin, Philip N. Moore, Prof. James F. Kemp, and Charles Kirchhoff. Telegrams of greeting were sent to "Uncle" John

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‡ Papers in manuscript form available for consultation.

Fritz, of Bethlehem, Pa., and Dr. R. W. Raymond, at Washington, Conn.

On Thursday afternoon, Nov. 1, the members and guests were taken by automobiles to the extensive plant of the National Carbon Co., where, through the courtesy of Mr. Henry A. Barren, the General Superintendent, and a number of active assistants, the various departments were visited and inspected.

Through the kindness of E. W. Kronbach of the National Carbon Co., the following brief description of the interesting visit is here given:

> The itinerary started with an inspection of the company's library, which contains practically all the up-to-date technical literature, magazines and books, which could in any way help its employees. The arc-lamp testing-laboratory was next visited, where several hundred arc-lamps, representing nearly every type of lamp in commercial use, were under test to prove the carbon electrodes. In the room above hundreds of dry cells were being tested in every conceivable way. A large number of these cells were tested merely to check the uniformity of the daily output, while others were experimental cells, made up by the Research Department, to improve, if possible, the present product.
>
> The factory was then visited and the manufacture of dry cells watched through the various processes of shaping and soldering the zinc cans, lining them, inserting the electrode tamping the mix, soldering on the binding-post, and finally pouring in the seal. The several careful testing-methods to which each cell is subjected to insure that only perfect cells should leave the factory were carefully explained and noted.
>
> The route then lay through the mixing- and grinding-rooms, where the various ingredients for the lighting-carbons, electrodes, brushes, and cells are thoroughly ground and mixed. From here the forcing-room, where all the arc-lamp carbons are forced through dies, was visited. In the upper end of the building the brushes and the dry cell electrodes are molded under enormous hydraulic pressure. The molding, forcing and stamping of the many carbon specialties, such as contacts, plungers, discs, and diaphragms of all sizes and shapes, created a great deal of interest. Next was visited the furnace-room, containing furnaces to bake the carbon products and impart the necessary electrical and physical properties.
>
> The balance of the work seen was more of a mechanical nature, and consisted in cleaning from the adhering packing-materials the carbon products, after coming from the furnaces. The lighting-carbons are then put through the sorting, gauging and coring processes, while the brushes are cut, planed, beveled and copper-coated, according to the demands of the trade. The various products are then wrapped, packed and shipped.
>
> In the last place visited—the brush-testing laboratory—are machines, both electrical and mechanical, to test the brushes thoroughly before shipping. The following tests are made: specific resistance, strength, hardness, density, friction-tests at various speeds, contact-resistance tests at different brush-pressures and current-densities.

A trip to the ore-docks of M. A. Hanna & Co. followed, and the party viewed with awe the impressive motions of the

Hewlett unloader, which was discharging with its massive 17-ton bucket the cargo of iron-ore from the S. S. *David B. Meacham.*

Special mention should be made of the attractive and interesting souvenir program prepared by the Program Committee under Mr. C. B. Murray, *Chairman*. This small volume, cloth-bound in attractive green, contains not only the data of officers, committees, titles and authors of papers, and other matters of routine importance; but also a brief illustrative description of the principal buildings, parks, and other public institutions for which Cleveland is noted,

The following list, doubtless incomplete, comprises the names of members and guests enrolled at Headquarters:

Abbott, Robert R., Cleveland, Ohio.
Ammon, Mark A., Cleveland, Ohio.
Applegate, J. S., Youngstown, Ohio.
Arluck, A., Cleveland, Ohio.
Backert, A. O., Cleveland, Ohio.
Badger, A. C., Cleveland, Ohio.
Bain, H. Foster, San Francisco, Cal.
Barren, Henry A., Cleveland, Ohio.
Barrows, W. H., Jr., Duluth, Minn.
Benjamin, E. H., San Francisco, Cal.
Bever, C. A., Cleveland, Ohio.
Blauvelt, W. H., Syracuse, N. Y.
Bowler, R. P., New York, N. Y.
Braid, Arthur I., New York, N. Y.
Braman, H. T., Youngstown, Ohio.
Brayer, M. S., Sharon, Pa.
Byrne, J. H., Cleveland, Ohio.
Callender, L. W., Cleveland, Ohio.
Carey, C. S., Cleveland, Ohio.
Chute, H. O., New York, N. Y.
Clagett, Thomas H., Bluefield, W. Va.
Coe, W. W., Roanoke, Va.
Collier, L. A., Cleveland, Ohio.
Collord, George L., Sharon, Pa.
Cook, Edgar S., Pottstown, Pa.
Corning, C. R., New York, N. Y.
Cox, Jennings S., Jr., Santiago de Cuba.
Crafts, W. N., Cleveland, Ohio.
Crowell, Benedict, Cleveland, Ohio.
Croxton, D. T., Cleveland, Ohio.
Croxton, Mrs. D. T., Cleveland, Ohio.
Croxton, S. W., Cleveland, Ohio.
Dalton, H. G., Cleveland, Ohio.
Danforth, A. E., Cleveland, Ohio.
D'Invilliers, E. V., Philadelphia, Pa.
Dunham, L. E., Cleveland, Ohio.
Dunham, Mrs. L. E., Cleveland, Ohio.
Emmerton, F. A., Cleveland, Ohio.
Emmerton, Mrs. F. A., Cleveland, Ohio.
Eustis, Augustus H., Boston, Mass.
Eustis, W. E. C., Boston, Mass.
Findley, A. I., New York, N. Y.
Ford, E. L., Youngstown, Ohio.
Fowler, D. D., Cleveland, Ohio.
Fruehauf, P. A., Cleveland, Ohio.
Fulton, Charles H., Cleveland, Ohio.
Fulton, H. F., Cleveland, Ohio.
Gayley, James, New York, N. Y.
Goodale, Stephen L., Pittsburg, Pa.
Gresham, A. L., New York, N. Y.
Griswold, C. L., Pittsburg, Pa.
Hale, H. C., Cleveland, Ohio.
Hamilton, J. H., New York, N. Y.
Hansell, N. V., New York, N. Y.
Harrison, H. T., Cleveland, Ohio.
Haynes, Elwood, Kokomo, Ind.
Heston, R. P., Toledo, Ohio.
Hibbard, Henry D., Plainfield, N. J.
Hindshaw, Henry H., Alpena, Mich.
Hoyt, W. H., Cleveland, Ohio.
Humphrey, George S., New York, N. Y.
Humphrey, Mrs. G. S., New York, N. Y.
Hunt, Robert W., Chicago, Ill.
Ireland, J. D., Duluth, Minn.
Ives, L. E., Cleveland, Ohio.
Jewitt, F. G., Minneapolis, Minn.
Johnson, J. E., Jr., Ashland, Wis.
Johnson, Mrs. J. E., Jr., Ashland, Wis.

Johnson, Nason, Alpena, Mich.
Kelly, William, Vulcan, Mich.
Kelly, Mrs. William, Vulcan, Mich.
Kemp, James F., New York, N. Y.
King, Paul H., Philadelphia, Pa.
Kirchhoff, Charles, New York, N. Y.
Kirchhoff, Mrs. Chas., New York, N.Y.
Knapp, G. F., Cleveland, Ohio.
Koehler, William, Cleveland, Ohio.
La Follette, H. M., La Follette, Tenn.
Laist, Frederick, Anaconda, Mont.
Lake, E. F., Cleveland, Ohio.
Lash, Horace W., Cleveland, Ohio.
Ledoux, A. R., New York, N. Y.
Lihme, I. P., Cleveland, Ohio.
Livingston, J. B., Cleveland, Ohio.
Longyear, E. J., Minneapolis, Minn.
Lyon, D. A., Pittsburg, Pa.
McKee, A. G., Cleveland, Ohio.
Macon, W. W., Brooklyn, N. Y.
Malcolmson, James W., Kansas City, Mo.
Marshall, S. W., Johnstown, Pa.
Marting, H. A., Ironton, Ohio.
Mather, Samuel, Cleveland, Ohio.
Mather, W. G., Cleveland, Ohio.
Menke, Paul O., Sharon, Pa.
Merry, H. G., Cleveland, Ohio.
Miller, L. B., Cleveland, Ohio.
Moore, Philip N., St. Louis, Mo.
Motter, W. D. B., Jr., Trenton, Ont.
Murray, C. B., Cleveland, Ohio.
Murray, Mrs. C. B., Cleveland, Ohio.
Parker, Edward W., Washington, D. C.
Parrock, H. P., Buffalo, N. Y.
Pecanka, William, Cleveland, Ohio.
Pelton, J. S., New York, N. Y.
Pfeiffer, George W., Santiago de Cuba.
Pierce, C. B., Cleveland, Ohio.
Pitkin, S. H., Akron, Ohio.
Pitkin, Mrs. S. H., Akron, Ohio.
Playter, Franklin, Boston, Mass.
Price, J. D., New Straitsville, Ohio.
Price, John M., Cleveland, Ohio.
Rand, Charles F., New York, N. Y.
Rand, Mrs. Chas. F., New York, N. Y.
Ranney, W. B., Lakewood, Ohio.
Rattle, William, Jr., Cleveland, Ohio.
Read, J. B., Cleveland, Ohio.
Reinhardt, G. A., Cambridge, Mass.
Reitz, J. C., West Park, Ohio.
Richards, F. B., Cleveland, Ohio.
Robinson, C. Snelling, Youngstown, Ohio
Roeber, E. F., New York, N. Y.
Sawyer, Willard N., Cleveland, Ohio.
Schoefer, C. A., Cleveland, Ohio.
Schumacher, Dr. W., Berlin, Germany.
Schniewind, Dr. F., Englewood, N. J.
Sheridan, R. B., Cleveland, Ohio.
Sherrerd, J. M., Easton, Pa.
Stillwagon, S. C., Cleveland, Ohio.
Smith, Albert W., Cleveland, Ohio.
Spilsbury, E. Gybbon, New York, N.Y.
Stay, T. D., Cleveland, Ohio.
Stebbins, H. S., Cleveland, Ohio.
Stoek, H. H., Urbana, Ill.
Stoughton, Bradley, New York, N. Y.
Stoughton, Mrs. Bradley, New York, N. Y.
Struthers, Joseph, New York, N. Y.
Sweetser, R. H., Columbus, Ohio.
Sweetser, Mrs. R. H., Columbus, Ohio.
Taylor, Knox, High Bridge, N. J.
Taylor, Mrs. Knox, High Bridge, N. J.
Textor, Oscar, Cleveland, Ohio.
Textor, R. B., Cleveland, Ohio.
Thropp, J. E., Jr., Indiana Harbor, Ind.
Thropp, Mrs. J. E, Jr., Indiana Harbor.
Treat, F. H., Cleveland, Ohio.
Vallat, B. W., Ironwood, Mich.
Van Horn, Frank R., Cleveland, Ohio.
Vogel, Felix A., New York, N. Y.
Volk, Karl E., West Park, Ohio.
Walther, Fred. F., Cleveland, Ohio.
Weld, C. M., Low Moor, Va.
Wellman, S. T., Cleveland, Ohio.
Wendel, Edmund, Cleveland, Ohio.
Whelan, M. T., Cleveland, Ohio.
Wood, C. L., Cleveland, Ohio.
Wood, Walter, Philadelphia, Pa.
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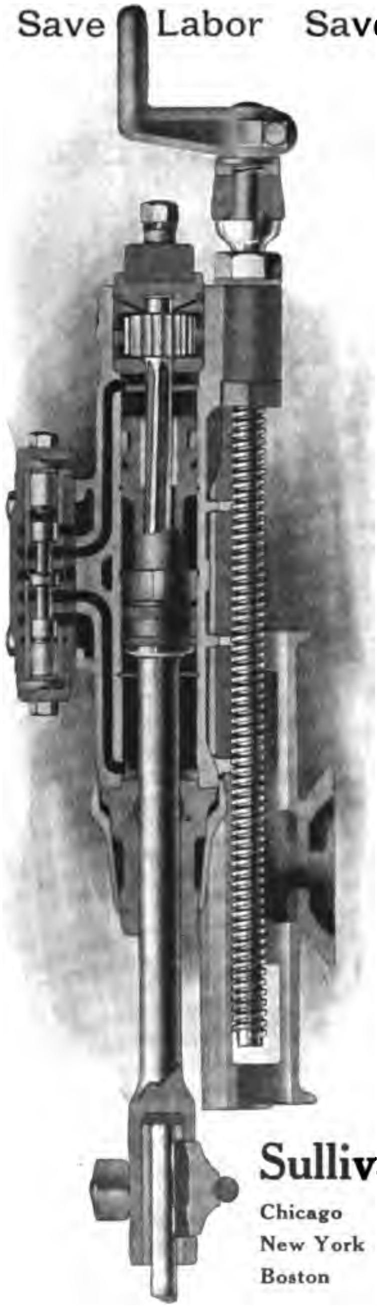

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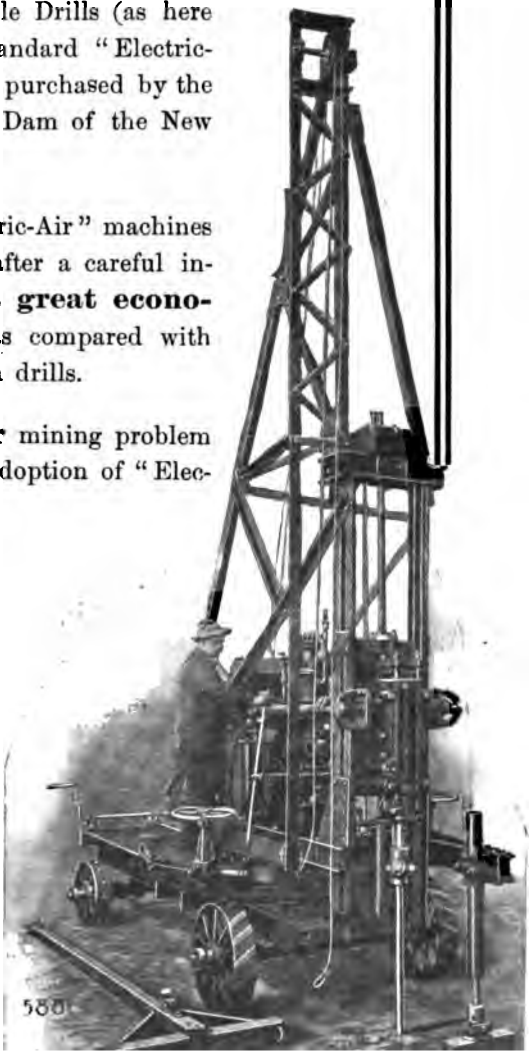

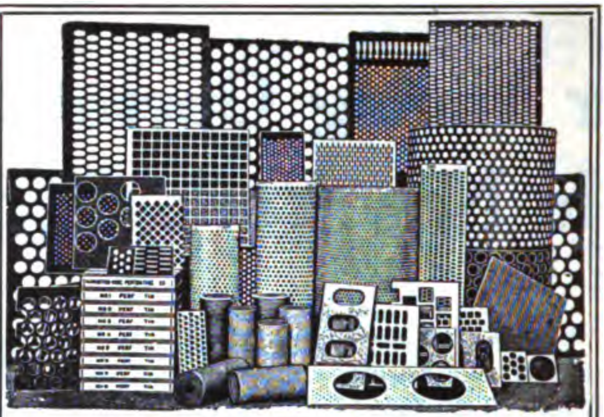

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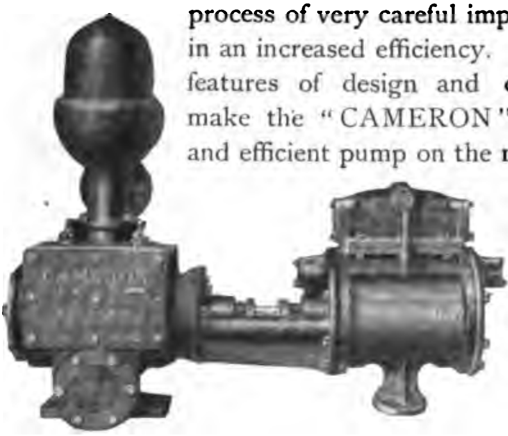

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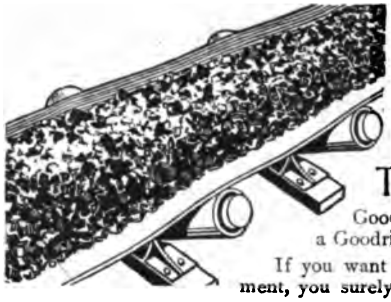

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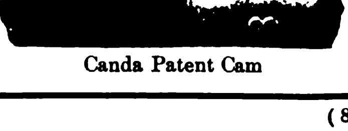

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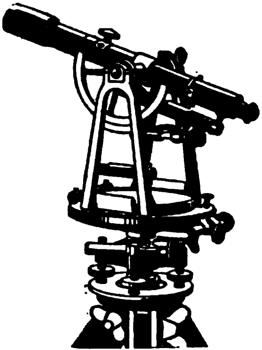

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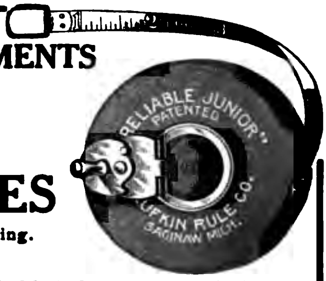
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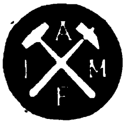

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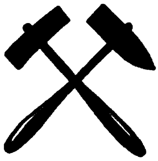

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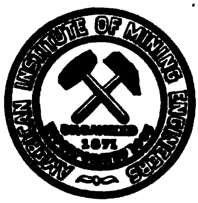
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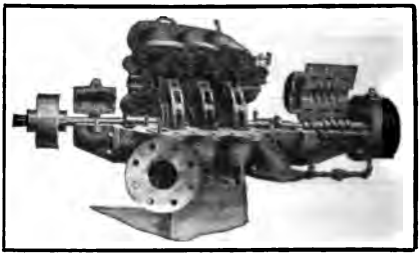

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CONTENTS:

SECTION I.—INSTITUTE ANNOUNCEMENTS. PAGE

List of Officers and Committees, ... ii
Institute Announcements, ... v
Library, ... xix
Membership, ... xxv

SECTION II.—PAPERS AND DISCUSSIONS.

The Microstructure of Iron and Steel. By William Campbell, ... 1357
Notes on the Metallography of Alloys. By William Campbell, ... 1381
The Action of Various Commercial Carbonizing-Materials. By Robert R. Abbott, . 1407
The Constitution and Melting-Points of a Series of Copper-Slags. By Charles H. Fulton, ... 1457
Recent Developments in the Inspection of Steel Rails. By Robert W. Hunt, . . 1487
The Concentration of Iron-Ores. By N. V. Hansell, ... 1497
A Graphic Solution of D'Arcy's Formula for the Transmission of Compressed Air in Pipes. By Nathaniel Herz, ... 1519
The Sulphatizing-Roasting of Copper-Ores and Concentrates. By Utley Wedge, . 1523
The Development of the Parkes Process in the United States. By Ernst F. Eurich, 1531
Discussion of Mr. Johnson's paper, The Effect of Alumina in Blast-Furnace Slags. By R. H. Sweetser, and Others, ... 1541
Discussion of Mr. Ammon's paper, Measurements and Relations of Hardness and Depth of Carbonization in Case-Hardened Steel. By J. E. Johnson, Jr., and Bradley Stoughton, ... 1553
Discussion of Mr. Howe's paper, Notes on Ruff's Carbon-Iron Equilibrium Diagram. By J. E. Johnson, Jr., and Others, ... 1555
Discussion of Mr. Stoughton's paper, Notes on Titanium, and the Cleansing-Effect of Titanium on Cast-Iron. By J. E. Johnson, Jr., and James F. Kemp, . . 1559
Discussion of Mr. Sweetser's paper, Blowing-In a Blast-Furnace. By J. E. Johnson, Jr., 1561
Discussion of Professor Sauveur and Mr. Reinhardt's paper, Notes on the Case-Hardening of Special Steels. By Robert R. Abbott, ... 1563
Discussion of Mr. Hansell's paper, The Concentration of Iron-Ores. By E. G. Spilsbury, and Others, ... 1565
Discussion of Mr. Abbott's paper, The Action of Various Commercial Carbonizing-Materials. By H. D. Hibbard, ... 1569
Discussion of Mr. Hunt's paper, Recent Developments in the Inspection of Steel Rails. By H. D. Hibbard, and Others, ... 1571
Index of Titles and Authors of the *Bulletin* for the Year 1912, ... 1575

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TABLE OF CONTENTS.

SECTION I. INSTITUTE ANNOUNCEMENTS.

SECTION II. TECHNICAL PAPERS.

[The American Institute of Mining Engineers does not assume responsibility for any statement of fact or opinion advanced in its papers or discussions.]

PAGE
No. 1. The Microstructure of Iron and Steel. By WILLIAM CAMPBELL, . 1357
No. 2. Notes on the Metallography of Alloys. By WILLIAM CAMPBELL, 1381
No. 3. The Action of Various Commercial Carbonizing-Materials. By ROBERT R. ABBOTT, ... 1407
No. 4. The Constitution and Melting-Points of a Series of Copper-Slags. By CHARLES H. FULTON, ... 1457
No. 5. Recent Developments in the Inspection of Steel Rails. By ROBERT W. HUNT, ... 1487
No. 6. The Concentration of Iron-Ores. By N. V. HANSELL, ... 1497
No. 7. A Graphic Solution of D'Arcy's Formula for the Transmission of Compressed Air in Pipes. By NATHANIEL HERZ, ... 1519
No. 8. The Sulphatizing-Roasting of Copper-Ores and Concentrates. By UTLEY WEDGE, ... 1523
No. 9. The Development of the Parkes Process in the United States. By ERNST F. EURICH, ... 1531
No. 10. Discussion of Mr. Johnson's paper, The Effect of Alumina in Blast-Furnace Slags. By R. H. SWEETSER, and OTHERS, ... 1541
No. 11. Discussion of Mr. Ammon's paper, Measurements and Relations of Hardness and Depth of Carbonization in Case-Hardened Steel. By J. E. JOHNSON, JR., and BRADLEY STOUGHTON, ... 1553
No. 12. Discussion of Mr. Howe's paper, Notes on Ruff's Carbon-Iron Equilibrium Diagram. By J. E. JOHNSON, JR., and OTHERS, . . 1555
No. 13. Discussion of Mr. Stoughton's paper, Notes on Titanium, and the Cleansing-Effect of Titanium on Cast-Iron. By J. E. JOHNSON, JR., and JAMES F. KEMP, ... 1559
No. 14. Discussion of Mr. Sweetser's paper, Blowing-In a Blast-Furnace. By J. E. JOHNSON, JR., ... 1561
No. 15. Discussion of Professor Sauveur and Mr. Reinhardt's paper, Notes on the Case-Hardening of Special Steels. By ROBERT R. ABBOTT, 1563
No. 16. Discussion of Mr. Hansell's paper. By E. G. SPILSBURY, and OTHERS, 1565
No. 17. Discussion of Mr. Abbott's paper. By H. D. HIBBARD, ... 1569
No. 18. Discussion of Mr. Hunt's paper. By H. D. HIBBARD, and OTHERS, 1571
No. 19. Index of Titles and Authors of the *Bulletin* for the Year 1912, . 1575

All communications concerning the contents of this Bulletin should be addressed to JOSEPH STRUTHERS, Ph.D., Secretary and Editor, 29 W. 39th St., New York, N. Y.

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Final Adjourned Annual Business Meeting.

The following abstract of the minutes of the final Adjourned Annual Business Meeting, held Nov. 12, 1912, in the office of the Institute, 29 West 39th Street, New York City, is here presented for the information of the membership at large.

On the announcement by the Secretary that there was a quorum present, President James F. Kemp called the meeting to order. The minutes of the Adjourned Annual Business Meeting of Oct. 7, 1912, were read and approved. (See p. xi,*Bulletin* No. 71, November).

Prof. Jos. W. Richards, in behalf of the Special Committee of the Board of Directors and the Council (Messrs. Richards, Kirchhoff, and Rand), withdrew the two proposed amendments to the Constitution offered by the said Committee at the Adjourned Annual Business Meeting of Oct. 7, 1912. (See *Bulletin* No. 70, October, 1912, p. vi., and Circular Announcement No. 8, Oct. 15, 1912, p. 15.)

Professor Richards presented a syllabus of the proposed amendments to the Constitution and By-Laws prepared by the Special Committee. Several members present discussed various sections of the proposed Constitution, and the Committee advised that the suggestions would be duly considered before submitting the final document, which would be duly presented and sent to the membership at large at least 30 days before the next Annual Business Meeting, Feb. 18, 1913.

Professor Richards presented a resolution, seconded by Mr. Kirchhoff, making a change in the present Certificate of Incorporation, so that it would conform to the proposed changes in the Constitution and By-Laws; said resolution was unanimously adopted.

Professor Richards presented another amendment to the Constitution, which had been purposely omitted from the printed list, but had been requested by many members, namely, the creation of a "Class of Fellows." This question was freely discussed at the Cleveland meeting (see *Bulletin* No. 71, November, 1912, p. 1348), and in view of the diversity of opinion it was decided to present it in separate form for separate action of the membership at large at the Annual Business Meeting, Feb. 18, 1913.

In answer to question, the President announced that the proposed amendments to the Constitution and By-Laws presented by Messrs. Corning and Stone at the Adjourned Annual Business Meeting, Oct. 7, 1912, and published in Circular Announcement No. 8, Oct. 15, 1912, were still on record.

Professor Richards presented a syllabus of the proposed amendments to the By-Laws, and several sections were discused by members present, but no action of record was taken.

Mr. Charles F. Rand read a letter from Mr. L. K. Armstrong, Secretary of the Spokane Local Section of the Institute, referring to the proposed amendments to the Constitution, and he announced that many of Mr. Armstrong's suggestions were, in one form or another, already incorporated in the proposed amendments submitted by the Special Committee.

A special vote of thanks was passed to Dr. Albert R. Ledoux for the valuable assistance rendered by him to the Special Committee in the preparation of the proposed amendments to the Constitution and By-Laws.

Iron and Steel Division.

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Albert Sauveur,
F. W. C. Schniewind,
A. A. Stevenson,
Felix A. Vogel,
Leonard Waldo,
William R. Webster.

At the invitation of William R. Webster, of Philadelphia, a number of prominent metallurgists, engineers of maintenance of way, and railroad officials met on the evening of Nov. 7 under the auspices of the Iron and Steel Committee of the Institute, at the headquarters, for the informal presentation of a paper by Benjamin Talbot, Managing Director of the Cargo Fleet Works, at Middlesbrough, England. It was the first occasion on which Mr. Talbot had personally described his process for producing solid steel ingots, recently developed, with special reference to ingots for rails and heavy sections of structural material.

Mr. Talbot analyzes the problem of segregation and of the formation of cavities and blow-holes in steel ingots and the influence of modern rail practice upon these defects. Mr. Talbot declared it to be his experience that in rolling rails the difference of size of ingots does not affect the question of segregation, cavities, or blow-holes. He holds that the use of very large ladle-heats is distinctly a step in the wrong direction, since very large nozzles must be used.

So far as the elimination of blow-holes is concerned, soundness in the ingots may be secured by the well-known powerful deoxidizers, aluminum, silicon, and ferro-titanium. They all produce solid steel except for the large central cavity. They all diminish segregation. Mr. Talbot therefore reasoned that if a deoxidizer such as aluminum were used to eliminate the blow-holes in the outer envelope of the ingot, and the ingot or the top portion thereof were reduced while the center was liquid, the central pipe would not form. Mr. Talbot has therefore adopted the following method of procedure: An ingot of at least 20 by 24 in. in cross-section is used and 2 oz. of aluminum to the ton of steel is added as the ingot is being poured. Aluminum is preferred as the deoxidizer because o its low cost and its low melting-point. It causes a perfectly sound outer envelope to be formed, and the metal solidifies earlier than it no deoxidizer were used. The ingot can therefore be stripped earlier. It is then put into the soaking-pit to allow the envelope to become thicker, and at the same time to attain a surface-temperature proper for compression. The 20 by 24 in. ingot is then reduced in the blooming-mill to about 18 by 18 in. section, while the center is still liquid. After this compression the ingot is returned to the soaking-pit for a proper heating and solidifying of the mass, and when this has been accomplished it is rolled down into a bloom, cropped, and passed to the rail-mill.

In analyzing and taking sulphur-prints from the face of the compressed ingot, which was cut longitudinally through its center, Mr.

Talbot made an interesting discovery. He found that when an ingot was compressed while the center was liquid, no segregation developed in the center of the upper part, as is usual, but that it was driven to the internal face of the solid envelope in a fairly regular percentage over the entire liquid area.

The solid outer envelope consists of the normal steel of the heat and is about 3 in. thick. The carbon in this portion is from 0.65 to 0.70 per cent. The carbon in the segregated inner skin is higher, being from 0.75 to 0.80 per cent., while the center is softer, with about 0.50 per cent. The sulphur and phosphorus also vary in these strata, but as the phosphorus is low in this steel, it was not considered.

The rail produced from the ingots so treated has the same characteristic formation as the squeezed ingot, viz.: a hard working-face, a harder ring at the back of it, and a softer center.

It is the question of this new formation which rail-manufacturers must discuss with railroad engineers and metallurgists. If they accept this new structure with the guarantee that with this formation they have no pipe in the rail, then it will be for manufacturers te consider the laying down of the necessary preparatory plant to accomplish this liquid compression of the ingot, as it cannot be properly accomplished in any existing rail-mill without largely decreasing the output. So far the rails produced by this method have successfully passed the drop test of the British Standard Specifications,

Mr. Talbot's paper was accompanied by a series of photographs and tables of analyses, which will form a part of the final paper to be printed at a later date. In the animated discussion which followed, Dr. Henry M. Howe, A. A. Stevenson, E. F. Kenney, Dr. P. H. Dudley, W. C. Cushing, Max H. Wickhorst, Henry D. Hibbard, Bradley Stoughton, and others participated.

New York Local Section.

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Meeting, Nov. 22, 1912.

The November meeting of the New York Section of the American Institute of Mining Engineers was held in the rooms of the Institute, 29 West 39th Street, Friday evening, Nov. 22, 1912. The meeting was called to order by the Chairman, Dr. George F. Kunz.

By invitation, Prof. J. W. Richards, Chairman of the Committee appointed by the Institute to prepare a revised Constitution and By-Laws, gave a brief synopsis of the changes which the Committee has adopted.

The speaker of the evening, Mr. N. H. Darton, Geologist of the U. S. Bureau of Mines, presented a paper entitled, The Structure of the Northern Anthracite Basin Relative to Form of Folds, which was illustrated by many lantern-slides. An abstract of this paper and the subsequent discussion follows:

The principal feature of this paper was a map showing the structure of the coal-basin by means of contour-lines. This map was prepared incidentally in connection with an extensive study of the origin of coal-gas and the geological conditions under which that gas occurs. The Northern anthracite-field was one of the areas selected for detailed investigation, and the structure had to be worked out in order to ascertain to what extent deformation was a factor in the gas-problem. The map was compiled from mine-maps so far as practicable, but in areas not reached by mining, the structure was determined from surface data, aided by numerous bore-hole records. The horizon selected for contouring was the lowest notable coal-bed (Dunmore-Red Ash) and its configuration is represented by 100-ft. contour-lines with sea-level datum. The contours based on mine-surveys in the lower beds are shown as full lines; those constructed from workings in overlying beds are shown by broken lines; while in areas not yet worked, dotted lines are used. The structure is still further represented by 15 vertical sections which cross the basin at frequent intervals.

Much attention was given to the extension of the different coal-beds and it is now practicable to correlate all the beds throughout the basin. The investigation has incidentally disclosed some very interesting structural details, showing notable discordance in the forms of the flexed beds at different depths and in harder and softer layers. It was necessary.to make a very careful study of the conditions under which variations of this character occur in order to represent the probable position of important coal-beds in areas not yet worked. As such a representation becomes a prediction and may affect plans for the future exploitation of coal, its accuracy is a matter of great economic importance, especially in the deeper basins where some of the coal lies nearly half a mile below the surface.

Discussion.

Prof. James F. Kemp indorsed Mr. Darton's paper, and said that the more familiar he becomes with the anthracite region the more he is impressed with the wonderful folding which has taken place.

Mr. William Griffith said that he and his associates had made 20 or 30 sections of the basin, which had resulted in considerable information. Furthermore, that the mining will probably develop curious conditions in the folding which cannot be foretold from surface indications or drill-holes.

Prof. Benjamin L. Miller stated that he and his associates are interested in a similar problem southeast of the Cambrian district in the limestone, and that a great many details similar to the ones presented by Mr. Darton are being discovered, such as examples of folds overturned in both directions.

Prof. Joseph W. Richards, in moving a vote of thanks, impressed upon the members present that all honor should be given to the men who go ahead and make approximations on the best information we have, and try to point out where things probably will be.

Dr. Arthur H. Elliott, in seconding the motion, dwelt upon the enormity of the work required of the engineers before the maps could be made. He also inquired as to the quality of the coal that is being mined at present: whether it is improving or getting poorer? Mr. Darton replied that the operators are trying to conserve the coal-fields, and are shipping as low a quality of coal as the market will take. When the market demands high-grade coal it can be delivered, as it is there in millions of tons.

Professor Richards also replied to Dr. Elliott's question: The best coal was mined on the Wyoming range, and now the Southern field, where the heavy pitching occurs, is being mined.

Mr. W. S. Ayres, who is thoroughly familiar with the preparation of coal, replied to Dr. Elliott as follows: In mining from the southwestern portion of the field, where the great pitches occur, all of the material which falls away from the roof must be taken out with the coal, resulting in a more complicated preparation than in the upper region. Veins 2 ft. thick are being worked, so that the whole process of preparation for the market is very expensive.

Dr. A. R. Ledoux remarked that many people are buying coal by a standard specification, which has been adopted and is coming more and more in vogue. Large consumers have their coal sampled and analyzed.

Mr. Ayres desired to know whether there was any relation between the width and depth of the basin relative to the degree of faulting. Mr. Darton replied that there is in general, and further discussed this point.

Announcement.

Before adjournment the Secretary announced that on Friday, Dec. 20, Dr. A. E. Barlow will deliver a paper on the Cobalt-Ores of Ontario.

Attendance.—The Secretary mailed 680 notices to the members of the New York Section and members of the Institute residing in the coal-fields of Pennsylvania. In reply, 43 members of the New York Section and 12 members residing in the coal-fields signified their intention of attending the meeting. The total number present was about seventy.

Budget.—Checks to the amount of $768 have been received and credited to the fund of the New York Section. Members who have not subscribed to this fund and are interested in the success of the Section are requested to send their checks to the Secretary of the Section. Most of the checks received have been for $5 or $10.

LOUIS D. HUNTOON, *Secretary-Treasurer*,
165 Broadway, New York, N. Y.

Boston Local Section.

ROBERT H. RICHARDS, *Chairman.*
ALBERT SAUVEUR, *Vice-Chairman.*
AUGUSTUS H. EUSTIS, *Secretary-Treasurer.*

The sixth meeting of the Boston Local Section of the Institute was held at the Technology Club, Monday evening, Nov. 4. 1912, about 20 members being present.

Chairman Robert H. Richards presided at the dinner, at the close of which the meeting was called to order, and Prof. Raphael Pumpelly spoke informally of his excavations in Turkestan.

Professor Pumpelly explained that he had always wanted to dig up humanity in the past, but had never had an opportunity until very recently; the delay in some respects was due to an illness with smallpox in Pekin more than 50 years ago. He said that he was there as a guest of the American Legation, and had been spending much effort studying the geology of China. He found that various articles that were in reality fossils could be collected from the bric-a-brac shops. He had also accumulated a large quantity of notes made from a Chinese Geology extending over 3,000 volumes. These notes had been collected for him by assistants, and he was able to make use of them, the fossils and other information during his convalescence, to establish one or more hypothetical geological cross-sections of the country. This he compared with a map of the Empire made by the Jesuits. In this work his attention was drawn to a number of lakes in the western part of the Empire, which had no outlets, and which were accordingly salt. In studying these he became convinced that they were originally a part of a great inland sea, and he was seeking for a connection between these and the cities which had been buried by sand. This study made him anxious to return to China later, and carry on active investigations, and arrangements for such a trip were finally made through the co-operation of the Carnegie Institute. In this trip he was accompanied by Professor Davis, so that he was able to divide his forces and cover a large amount of territory. The problem, which had at first seemed simple and easy, appeared much more difficult as he approached the actual scene of his operations, and it seemed at times that his travels would be without results. During this work of reconnaissance, he was aided in every way by officials of the Russian government, who ordered that his train should be stopped at his pleasure; he was also supplied with horses for side-excursions. During these travels, his attention was called to two or more mounds in the great desert, which rose 20 ft. or more above the level of the desert, and on his way home he determined to investigate these more closely. One of them had been opened by a Russian General, who was hunting treasure. The section was stratified, full of pottery and bones, and he recognized at once that in these mounds was his problem. He made his plans to return the next year and carry on active excavations, but the Russian government, which had rendered him every assistance in his travels, was very loath to grant permission for him to do excavation-work. This was, however, finally secured, and he arranged with Doctor Schmidt, an archæologist, to take part in the work.

The work of excavation was commenced by sinking large pits, from 15 to 30 ft. in diameter, through the mounds and in the surrounding plain. It was like sampling a mine-prospect. In sinking these pits great care was taken to keep the floor level, and to collect and carefully label articles of interest. To this end two baskets were kept in each pit; one for bones, and the other for pottery and anything else that could be found. From these excavations it was found that pottery and bones were found to a depth of 20 ft. below the existing plain, and this was established as the base of the first civilization that had occupied this region. It was 45 ft. from the base of this civilization to the top of the mounds, and in this space three civilizations were found to be represented. Between the first and second civilizations was a sharp and well-defined partition, which was marked by a layer of burned and charred earth. In the first 10 ft. from the bottom up were found only bones of wild animals, which were recognized by Durst, an anatomist from Zurich, as the bones of a small horse, a wild sheep, a pig, and an ox. Above the first 10 ft. the bones began to show evidences of domestication, as indicated by their becoming, except in the case of the horse, more porous and less hard. At the beginning of the second civilization above the charcoal, there were found the remains of a hornless breed of sheep, which has been identified with a certain sheep found in the lake-dwellings in Switzerland, and of which there still exists one herd in the Alps. These sheep were associated with a round-headed man, which is of great interest as marking the discovery of the Asiatic origin of European domestic animals. The horse is a light-boned animal, evidently accustomed to roam great distances, and of great speed. It is named for Professor Pumpelly.

It was found that the second civilization had the same curious custom as the first, of burying their children under their hearth. These civilizations had some ornaments, which were mostly beads; in the case of the first civilization, turquoise; in the case of the second civilization, lapis lazuli. The second civilization also had copper ornaments with some elaborate design. The copper was pure. There was little arsenic, antimony, or lead, not more than 0.5 or 0.25 per cent. of tin, and this only in a few objects which were evidently not made for cutting; and it was, therefore, apparent that the art of making bronze was not then known, but this art of making bronze was known 2,200 years B. C. in Egypt, and what was known in Egypt was also known in Babylon, and must also have been known in the region of these excavations. It is apparent, therefore, that the second civilization had passed away before 2,200 years B. C.

Professor Pumpelly said that from the very bottom up, there were found pieces of large heavy pottery jars, crudely made and full of cavities. These cavities were found to be the casts of barley and wheat. It was thus seen that from the very beginning these civilizations lived on agriculture. In the second civilization there was found a weight, which is believed to be the parent of the Egyptian weight, known as a talent, about 13 kg. 600 g. The weight found weighed just under 13 kg., which small difference would easily be accounted for by wear, and inaccuracies in weight.

In the third civilization, which was found in a second mound to the south of the first one, there began to be used implements of iron. It seemed that the first civilization was established while the elevation of the country as a whole was rising; that the whole country was then washed away, and the second civilization established during the second rise and subsequent gradual depression of the country. During the third civilization the country began to rise again, and this civilization flourished into modern times, or about 80 years ago. It was a difficult matter to establish the relative ages of these different civilizations, but it seems fair to assume that the waste material with which they are buried grows in depth at a uniform rate. In order to study these matters further, Professor Pumpelly explained that he had gone to Egypt to study the mounds and excavations there, and he found very much the same conditions as in these mounds in Asia. He found in a city in Asia which was abandoned only 80 years ago, a mosque, on which was inscribed the date of 1440. The dirt and accumulation in this mosque from this time up to the present had amounted to about 9 ft., from which it would appear that the rate of growth was about 2.5 ft. per century; but in this mosque the earth was loose, whereas in the mounds the earth was hard and densely packed, and it seemed reasonable to assume, therefore, that the time represented by the earth in the mounds would correspond to perhaps 2 ft. per century. In Egypt more data is available, and the rate of growth has been established at from 1.3 to 1.8 ft. per century, from which Professor Pumpelly has calculated that the first civilization, which he has had the pleasure of revealing at Anau, began 8,000 years B. C.

A. H. EUSTIS, *Secretary-Treasurer*,
131 State St., Boston, Mass.

Puget Sound Local Section.

CHESTER F. LEE, *Chairman*.
JOSEPH DANIELS, *Secretary-Treasurer*.

The November meeting of the Puget Sound Local Section of the American Institute of Mining Engineers was held in the rooms of the College Club, Seattle, Wash. A. R. Jayne, of Seattle, Wash., presented a paper entitled, The Evolution of the Chilean Mill, which was well received.

The next meeting of the Section is called for Dec. 14, 1912.

JOSEPH DANIELS, *Secretary-Treasurer*,
University of Washington, Seattle, Wash.

Proceedings of the 103d Meeting of the Institute.

(Cleveland, Ohio, October, 1912.)

[SUPPLEMENTARY NOTE.]

[SECRETARY'S NOTE.—The following brief description of the works of the American Steel & Wire Co., which were visited by the Institute party Oct. 30, 1912 (see November *Bulletin*, No. 71, p. 1352), was received through the courtesy of General Superintendent Bertram D. Quarrie. Unfortunately, these notes were not in time to be incorporated in the Proceedings of the meeting, which was published in the November *Bulletin*, No. 71.—J. S.]

Visit of the American Institute of Mining Engineers to the Blast-Furnaces, Steel-Works, and Newburg Wire-Mill of the American Steel & Wire Co., Oct. 30, 1912.

The furnaces are located on the Cuyahoga river about 4 miles from the mouth. The steel-works and wire-mill are located about 7 miles from the furnaces, in a section of the city called Newburg, and are connected to the furnaces by a direct line of the Newburg & South Shore railway.

The production of the steel-works is billets and rods; while the wire-mill is a merchant mill, producing bright and coarse wire, wet coarse wire, mattress-wire, fine wire, straight and cut wire, and turned shafting, with galvanizing and tinning departments.

The members of the Institute met with representatives of the American Steel & Wire Co. at the B. & O. depot at 1.30 p.m., and were taken from there to the furnaces over the tracks of the B. & O. railroad by means of a special train. After visiting the furnaces, they boarded the train, which had been transferred to the tracks of the Newburg & South Shore, and were taken to the steel-works, passing *en route* the plants of the Corrigan-McKinney Co., the Cleveland Furnace Co., and the Cuyahoga works of the American Steel & Wire Co. From the steel-works they went to the wire-mill, just across the tracks of the Pennsylvania railroad. After completing the trip through the wire-mill they boarded special street-cars and were taken down town.

The ore-docks at the furnaces have a frontage of about 3,200 ft. and a storage capacity of about 750,000 tons. The ore is received by means of boats from the Lake Superior district, and unloaded at the furnace docks by means of four Hoover & Mason unloaders and bridges and two Hulett unloaders working in connection with one Wellman-Seaver bridge. These machines are all electrically driven, the Hoover & Mason having a capacity of 5 tons per bucket, and the Hulett a capacity of 10 tons. For conveying to the furnace-trestle bins the ore is loaded directly from boats into cars, or from stock-piles to cars. All of the material used at the furnaces, however, is handled from trestle-bins to larries and skip-cars below.

The furnaces are four in number and are located in a line parallel with the docks. They have an average capacity per furnace per 24 hr. of about 480 or 490 tons. Three of them are of the ordinary construction, viz., thick-lined walls, while the other is of thin-lined construction, having only 12 in. of brick in the stack-lining, the shell being cooled by the over-lapping water-bucket system.

The hoists to the furnaces are three steam-driven Otis hoists and one electrically-driven Otis hoist controlled by means of their magnet regulators. The blowing-engines, which were made by the Allis-Chalmers Co., are of the steam- and gas-driven types, and are located in different houses. The steam-engines are seven in number and are compound condensing, blowing an average of 750 cu. ft. of air per revolution. These engines are operated so that the output for two of them will blow one furnace. The gas-engines are two in number and are twin tandem, the gas-cylinders being 44 by 60 in. and the air-cylinders 80 by 60 in. The air-cylinders are direct connected and are of the Slick type, with a positive discharge-valve. These engines have a capacity of 670 cu. ft. of air per revolution, and are so operated that one engine will blow a furnace. The usual operation, however, is to have one steam-engine with its delivery split between two furnaces operating in connection with the gas-engines on the same two furnaces.

There are two boiler-houses; one containing 16 Cahall boilers with a capacity of 4,000 h-p., the other containing 16 Stirling boilers with a capacity of 4,800 h-p.

The surplus gas generated by the furnaces is used to drive four Allis-Chalmers gas-engine units having a capacity of 1,000 kw. each. The current is alternating, three-phase, 25-cycle, 13,200 volts. The output of the station is 1,750,000 to 2,000,000 kw. per month. Most of the current generated is transferred under this high tension to the Cuyahoga wire-works, which is electrically driven, and consumes about 1,000,000 kw-hr. per month. The balance is distributed between the motor-generator set and motors at the furnaces, the slag-crusher, and the Newburg wire-mill.

All of the iron produced at the furnaces is transferred to the steel-works molten in 16-ton ladle-cars during the week, and is taken care of on Sundays by casting over two strands of the pig machine.

The steel-mill has four departments: Bessemer converting-mill, open-hearth, blooming-mill, and the rod-mill.

The Bessemer converting-department has two mixers operated hydraulically and having a capacity of about 200 tons each. The pig-metal is delivered to the mixer-houses and lifted to the receiving-spout of the mixers by means of two 40-ton Morgan cranes.

There are four cupolas in line with the mixers, each having a capacity of about 250 tons per 24 hr. The blast for these cupolas is supplied by Sturtevant blowers. The metal from these is run to the mixers by means of a separate track and poured into the mixers by the same cranes as handle the pig-metal. The ladle which conveys the metal from the mixers to the converters is operated on a track at the same level with a pouring-spout into the converters, and is conveyed from the mixers to the converters by means of a rope-haulage.

These converters are two in number, having a capacity of 20 tons each. The blast for these converters is supplied by two Southwick blowing-engines. The converters are tipped hydraulically and the metal poured into a ladle, which is handled by a hydraulic jib-crane. This crane, in turn, swings to the pouring-platform, where the steel is poured into the ingot-molds, giving an ingot of approximately 6,000 lb. The ingots are then handled by means of a narrow-gauge engine to the stripper, either at the 38-in. or the 35-in. mill.

The open-hearth plant consists of six 60-ton stationary furnaces. All of the metal is supplied to these furnaces cold, and the furnaces are charged by two Wellman-Seaver charging-cars, the charging-boxes being brought to the charging-floor by means of broad-gauge engines. There are 14 Duff gas-producers. The fuel used is either producer-gas, or natural gas if available. The metal, after it is poured from the furnaces, is handled by a 75-ton Morgan crane. The production from this plant, in the form of ingots, is transferred to one of the blooming-mills.

The blooming-mills are 38- and 35-in., and are located in separate buildings. After the ingots have been stripped they are put into the soaking-pits by means of small-capacity electric cranes. In the 38-in. mill the soaking-pits are 20 in number, having a capacity of four ingots each. Each ingot is transferred from the pit to the transfer-car, which is operated by a rope-haulage. This car places the ingot on the end of the blooming-mill table. The blooming-mill is of a two-high type and blooms down to 7 by 8 in. The mill is driven by a duplex simple horizontal condensing engine. After passing into the shears the bloom is transferred by means of a transfer-table to the 23-in. billet-mill, which runs parallel to the blooming-mill. The billet-mill is three-high and is driven by a tandem compound horizontal condensing-engine. The billets are 4 by 4 in. in size, and are conveyed from the last table to the shears, where they are sheared to the proper length and dropped on to two chain-conveyors with dogs. These conveyors transfer the billets from the shears to the loaders. Loaders are of the chain type.

The general method of handling ingots at the 35-inch blooming-mill is the same as at the 38-in. mill. The soaking-pits, however, are only five in number, and each pit has a capacity of eight ingots. The mill is of the McIntosh-Hemphill two-high type, producing billets of a desired size. The method of drive for this mill is by compound condensing, horizontal reversing-engines.

The rod-mill is a four-train Garrett mill, the trains being one 18-in., one 12-in., and two 9-in. Heating-furnaces are two continuous Laughlin, and are fired by either producer- or natural gas. Hughes producers are being installed at the present time to replace the old type Wellman producers now in use. All of the product from this mill is in the form of coiled rods, and is carried by an overhead continuous conveyor by means of hooks into which the coils of rods are hung. This conveyor transfers the metal from the rod-mill across the Pennsylvania tracks to the Newburg wire-mill, where it is used in the finishing departments or loaded for shipment.

Kelvin Memorial Fund.

Acting for the Kelvin Memorial Committee, the Secretary-Treasurer announces that up to Dec. 9, 1912, subscriptions have been received from 92 members of the American Institute of Mining Engineers aggregating $497.75. Of this amount £100 1s. 6d. ($487.08) has been transmitted to J. H. T. Tudsbury, Honorary Secretary of the London Committee. The remaining subscriptions, together with others received up to the end of the present year, will be forwarded in due course.

Other National Engineering Societies interested in this good work are, the American Society of Civil Engineers, American Institute of Electrical Engineers, American Society of Mechanical Engineers, Canadian Society of Civil Engineers, and the Society of Naval Architects and Marine Engineers. The membership of these societies has been circularized, and very satisfactory contributions have been reported.

Emmons Volume on Ore-Deposits.

Ore-Deposits—a continuation of the "Posepny" Volume.

Comprising papers descriptive of ore-deposits and discussions of their origin, edited, with an introduction, by Dr. S. F. Emmons. The volume contains also a Biographical Notice of Dr. Emmons by his associate and friend, Dr. George F. Becker, and a comprehensive Bibliographical Index of the Science of Ore-Deposits, prepared by Prof. John D. Irving, of the Sheffield Scientific School of Yale University. Dr. Emmons had finished his editorial work and written his Introduction before his lamented death in 1910; and the Volume contains his last words upon the subject to which he had given the work of his life, and on which he was justly regarded as the foremost authority.

Contents.

Genesis of Certain Ore-Deposits. By S. F. Emmons.
Structural Relations of Ore-Deposits. By S. F. Emmons.
Geological Distribution of the Useful Metals in the United States. By S. F. Emmons. Discussion, by John A. Church, Arthur Winslow, S. F. Emmons, and William Hamilton Merritt.
Torsional Theory of Joints By George F. Becker. Discussion, by H. M. Howe, R. W. Raymond, C. R. Boyd, and George F. Becker.
Allotropism of Gold. By Henry Louis.
Superficial Alteration of Ore-Deposits. By R. A. F. Penrose, Jr.
Some Mines of Rosita and Silver Cliff, Colorado. By S. F. Emmons.
Genesis of Certain Auriferous Lodes. By John R. Don. Discussion, by Joseph Le Conte, S. F. Emmons, G. F. Becker, Arthur Winslow, W. P. Blake, and J. R. Don.
Influence of Country-Rock on Mineral Veins. By Walter Harvey Weed.
Igneous Rocks and Circulating Waters as Factors in Ore-Deposition. By J. F. Kemp.
Consideration of Igneous Rocks and Their Segregation or Differentiation as Related to the Occurrence of Ores. By J. E. Spurr. Discussion, by A. N. Winchell.
Chemistry of Ore-Deposition. By Walter P. Jenney. Discussion, by John A. Church.
Ore-Deposits near Igneous Contacts. By Walter Harvey Weed. Discussion, By W. L. Austin.
Ore-Deposition and Vein-Enrichment by Ascending Hot Waters. By Walter Harvey Weed.
Basaltic Zones as Guides to Ore-Deposits in the Cripple Creek District, Colorado. By E. A. Stevens.
Geological Features of the Gold-Production of North America. By W. Lindgren. Discussion, by W. G. Miller, and W. L. Austin.
Osmosis as a Factor in Ore-Formation, By Halbert Powers Gillette.
Ore-Deposits of Sudbury, Ontario. By Charles W. Dickson.
Genesis of the Copper-Deposits of Clifton-Morenci, Arizona. By W. Lindgren.
Copper-Deposits at San Jose, Tamaulipas, Mexico. By J. F. Kemp.
Magmatic Origin of Vein-Forming Waters in Southeastern Alaska. By A. C Spencer.
Genetic Relations of the Western Nevada Ores. By J. E. Spurr.
Are the Quartz Veins of Silver Peak, Nevada, the Result of Magmatic Segregation? By J. B. Hastings.
Occurrence of Stibnite at Steamboat Springs, Nevada. By W. Lindgren.
Summary of Lake Superior Geology with Special Reference to Recent Studies of the Iron-Bearing Series. By C. K. Leith.
Geological Relations of the Scandinavian Iron-Ores. By H. Sjögren.
Formation and Enrichment of Ore-Bearing Veins. (With Supplementary Paper.) By George J. Bancroft.
Distribution of the Elements in Igneous Rocks. By H. S. Washington.
Agency of Manganese in the Superficial Alteration and Secondary Enrichment of Gold-Deposits of the United States. By W. H. Emmons.
Cognate Papers.
Bibliography of the Science of Ore-Deposits. By J. D. Irving.

The volume will contain about 1,000 pages, and it is expected to be off the press early in 1913. Further details concerning time of publication, price, binding, etc., will follow in due course.

John Fritz Medal Award.

The award for the year 1912 of the John Fritz Medal Board of Award was made to Robert Woolston Hunt, of Chicago, Ill., " for his contributions to the early development of the Bessemer process."

The public presentation of the medal was made in the main auditorium of the Engineering Societies Building on Thursday evening, Dec. 5, 1912, and was followed by a reception to Mr. Hunt in the assembly rooms in the same building under the auspices of a committee representing the American Institute of Mining Engineers, the American Society of Mechanical Engineers, and the American Institute of Electrical Engineers acting as hosts.

Prof. James F. Kemp, the present President of the Board or Award, presided over the ceremonies, and Mr. John A. Brashear made the speech of presentation, supplemented by Mr. E. Gybbon Spilsbury, who delivered the medal and the diploma. A fitting response was made by Mr. Hunt. Dr. Henry S. Drinker, President of Lehigh University, delivered a special message of greeting from " Uncle John " Fritz, who was unable to be present.

The following awards of the John Fritz Medal have been made since the establishment of the Board:

1905.—Lord Kelvin.
1906.—George Westinghouse.
1907.—Alexander Graham Bell.
1908.—Thomas Alva Edison.
1909.—Charles T. Porter.
1910.—Alfred Noble.
1911.—Sir William Henry White.
1912.—Robert Woolston Hunt.

Special Notice.

The *Bulletin* is now entered at the Post Office at Second-Class Postage rate of one cent per pound, and in order to preserve this privilege it is necessary that the dues of members be paid within four months of January 1 of each year. If the dues are not paid within the period mentioned, a member's name must be removed from the regular subscription-list and the *Bulletin* mailed at the transient second-class postage rate of one cent for each four ounces or fraction thereof, prepaid by stamps affixed. It is therefore earnestly requested that dues be paid promptly—otherwise the Institute will be put to additional expense of postage and to added labor in removing and replacing names from the regular list, and in maintaining an additional separate mailing-list.

Library Research-Work.

The attention of members of the Institute is again directed to the research-work done by the librarian and his assistants, which should attract special attention from those members who have no access to the literature of subjects in which they may be interested.

During the year 1911 there were 143 searches made for members and non-members of the Founder Societies, and copies of the references have been preserved for the use of others. This work has been largely based on requests sent in by mail, from Japan, South Africa, Mexico, Canada, and England, as well as from different parts of the United States. The Librarian is confident that if it were more widely known that the library is equipped to undertake researches, the demand would increase beyond the ability of the present force to handle it. The library receives more than 700 technical periodicals which are available through the indexes for this special purpose.

Back Volumes of the Transactions.

The Board of Directors has authorized the following offers of sets of back volumes of the *Transactions*, at considerably reduced prices, to Members, Libraries, and Scientific Societies:

		Per Set.
I.	Five volumes, bound in half-morocco, from No. 36 (1906) to No. 40 (1910), ...	$20
II.	Ten volumes, bound in half-morocco, from No. 31 (1902) to No. 40 (1910), including Mexican Volume, . .	35
III.	Twenty volumes, bound in half-morocco, from No. 21 (1893) to No. 40 (1910), ...	50
IV.	Thirty volumes, bound in half-morocco, from No. 11 (1883) to No. 40 (1910), ...	60
V.	Thirty-nine volumes, bound in half-morocco, from No. 1 (1873) to No. 40 (1910), with the exception of No. 10 (1882), but including index for Volumes Nos. 1 to 35, and Nos. 36 to 40, ...	75
VI.	Nine volumes, bound in half-morocco, from No. 1 (1873) to No. 9 (1881), ...	25

Applications should be addressed to Joseph Struthers, Secretary, 29 West 39th Street, New York, N. Y.

The Emmons Research Fellowship of Economic Geology.

The Committee named below has been formed by friends of Samuel Franklin Emmons, late of the United States Geological Survey, to consider the best method of perpetuating his name. It has been decided that the memorial to him shall take the shape of a Research Fellowship, to be known as the Samuel Franklin Emmons Research Fellowship of Economic Geology, which is to be administered by Prof. James F. Kemp, of Columbia University, New York. Subscriptions are invited by his friends to this fund, which the Committee has fixed at $25,000.

Members of the Institute who desire to contribute to this fund will please communicate with the Treasurer, Benjamin B. Lawrence, 60 Wall Street, New York.

The Committee consists of the following:

GEORGE OTIS SMITH, Director, U. S. Geological Survey, Washington, D. C.
H. L. SMYTH, Harvard University, Cambridge, Mass.
JAMES DOUGLAS, 99 John Street, New York, N. Y.
J. A. HOLMES, Director, Bureau of Mines, Washington, D. C.
JAMES F. KEMP, Columbia University, New York, N. Y.
F. W. BRADLEY, San Francisco, Cal.
J. PARKE CHANNING, 42 Broadway, New York, N. Y.
SEELEY W. MUDD, 1001 Central Building, Los Angeles, Cal.
D. W. BRUNTON, Denver, Colo.
H. FOSTER BAIN, 420 Market Street, San Francisco, Cal.
T. A. RICKARD, London, England.
B. B. LAWRENCE, 60 Wall Street, New York, N. Y.

How to Use the "Transactions" of the Institute.

Buy a copy of the Complete Analytical and Alphabetical Index of Volumes I. to XXXV., inclusive; also the new Index of Volumes XXXVI. to XL.

Whether you do or do not own a full set of the *Transactions*, these Indexes will make all of the material contained in the forty volumes available at once without detailed research into each volume separately. Moreover, an easy search will show what particular papers you need to know more about, and perhaps to study. Thus, any person possessing these Indexes can ascertain at once what has been published in the *Transactions* on a given question, and can learn, by writing to the Secretary, what is its nature, whether it is still to be had in pamphlet form, where it can be consulted in a public library, at what cost it can be copied by hand, etc., etc.

In short, to those who own complete sets of the *Transactions*, these Indexes will be a great convenience; but to those who do not, they will be a professional necessity.

The Index Volumes I. to XXXV. is an octavo of 706 pages, containing more than 60,000 entries, duly classified with sub-headings, and including abundant cross-references. The limited edition is becoming exhausted. The new Index, Volumes XXXVI. to XL., supplementing the Index Volumes I. to XXXV., brings the classified references up to the date of Volume XL., June, 1910. Prices: Index Volumes I. to XXXV., bound in cloth, $5; bound in half-morocco, to match the *Transactions*, $6. Index Volumes XXXVI. to XL., bound in cloth, $1.50; bound in half-morocco, $2.50. The delivery charges will be paid by the Institute on receipt of the above price.

LIBRARY.

AMERICAN INSTITUTE OF ELECTRICAL ENGINEERS.
AMERICAN SOCIETY OF MECHANICAL ENGINEERS.
AMERICAN INSTITUTE OF MINING ENGINEERS.
UNITED ENGINEERING SOCIETY.

WILLIAM P. CUTTER, Librarian.

The libraries of the above-named Societies are open from 9 A.M. to 9 P.M. on all week-days, except holidays, from September 1 to June 30, and from 9 A.M. to 6 P.M. during July and August.

The Library contains about 50,000 volumes, including sets of technical periodicals and the publications of scientific and technical societies.

The members of the Institute, with few exceptions, are by the very nature of their profession forced to spend a large portion of their time in localities isolated from sources of information. To such members the Library can render valuable service through correspondence, and letters requesting information will receive special attention. The Library is prepared to furnish references and copies of articles on mining and metallurgical subjects; to determine, if possible, the existence of mining-maps, and to furnish general information as to the geology and mineral resources of all countries as far as these resources are known and published.

It is hoped that the members of the Institute will avail themselves freely of this special service. The Library will welcome inquiries on engineering subjects, and furnish information as far as such information is to be obtained.

All communications should be made as definite as possible so that the information received may be what is desired and not include collateral matter which may not be of interest. In this way the time spent in searching for such collateral matter will be saved, and as a result the information will be sent more promptly and in more usable shape.

The members of the Institute can be of service to the Library by forwarding copies of mining-reports, maps privately issued, and similar material, which will be classified, indexed, and made available to other members.

Suggestions for additions to the Library, either by purchase or personal solicitation as gifts, will be welcomed. It is hoped that members while in the city will use the Library freely, and assurance is given that most careful service will be rendered to them.

Library Accessions.

Nov. 1 to Nov. 30, 1912.

[Copies of the list of additions to the Libraries of the American Society of Mechanical Engineers and the American Institute of Electrical Engineers can be obtained on application to the Secretary of the American Institute of Mining Engineers.]

AMERICAN FERTILIZER HAND BOOK, 1912. Philadelphia, Ware Bros. Co., 1912. (Purchase.)

AMERICAN INSTITUTE OF MINING ENGINEERS. Souvenir of the One Hundred and Third Meeting, Cleveland, Oct. 28–31, 1912.

AMERICAN IRON AND STEEL ASSOCIATION. Annual Statistical Report for 1911, part II. Philadelphia, 1912. (Exchange.)

DIE ANLAGE UND DER BETRIEB DER EISENHÜTTEN. Vols. 1–3. By E. F. Dürre. Berlin, 1882. (Purchase.)

——— Atlas. Vols. 1–2. Berlin, 1882. (Purchase.)

CANADIAN PEAT SOCIETY. An Address delivered by Eugene Haanel at their Fourth Annual Meeting, July 25, 1910. Ottawa, 1910. (Gift of Canadian Peat Society.)

CARBONIC ACID INDUSTRY. By J. C. Goosmann. Chicago, 1907. (Purchase.)

CHEMICAL ARITHMETIC AND CALCULATION OF FURNACE CHARGES. By Regis Chauvenet. Philadelphia–London, 1912 (Purchase.)

CHEMICAL TECHNOLOGY AND ANALYSIS OF OILS, FATS AND WAXES. 3 vols. Ed. 4. By J. Lewkowitsch. New York, Macmillan Co., 1909. (Purchase.)

COMMERCIAL NOMENCLATURE. English-Spanish-Portuguese. Published by recommendation of the International American Conference, Aug., 1897. Washington, 1897. (Purchase.)

CONGRESO CIENTIFICO (1° PAN AMERICANO). Ciencias Naturales, Antropologicas y Etnologicas. Tomo III. (Volumen XVII.). Santiago de Chile, 1912. (Gift of Congreso Cientifico, 1° Pan Americano.)

ELECTRICAL MINING. Vol. 1, Nos. 1, 3–9; Vol. 2, Nos. 8, 12; Vol. 3, Nos. 6–12; Vol. 4, Nos. 1–12; Vol. 5, Nos. 1–4; Vol. 6, No. 1; Vol. 7, No. 2. Chicago, 1904–1910. (Gift of Electrical Mining Publishing Co.)

ELECTRICAL MINING INSTALLATIONS. By P. W. Freudemacher. New York, 1911. (Purchase.)

L'EXPLOITATION DES MINES DE COMBUSTIBLES, RECUEIL DES TEXTES OFFICIELS RELATIFS. By Louis Aguillon. Paris, 1912. (Purchase.)

DIE EXPLOSIVSTOFFE. Pts. 2–5. By Richard Escales. Leipzig, 1905, 1908–10. (Purchase.)

DIE FLÜSSIGEN BRENNSTOFFE IHRE GEWINNUNG EIGENSCHAFTEN UND UNTERSUCHUNG. By L. Schmitz. Berlin, Springer, 1912. (Purchase.)

GEOLOGICAL MAP OF A PORTION OF WEST TEXAS. By B. F. Hill and J. A. Udden. Austin, 1904. (Exchange.)

I GRANDI TRAFORI ALPINI FRÉJUS, SAN GOTTARDO, SEMPIONE ED ALTRE GALLERIE. (The Great Alpine Tunnels, Fréjus, San Gothard, Simplon and others.) By G. B. Biadego. Milano, U. Hoepli, 1906.

——— Atlante di 30 tavole. Milano, 1906. (Gift of W. L. Saunders.)

[NOTE.—This splendid volume, with its accompanying atlas of 30 plates, is not only a record of the work done in the successive great tunnels through the Alps, but also a critical history of the progress in the art of tunneling which those enterprises illustrate. Indeed, it might be said that they caused and constituted that progress. In this department, the world has followed the initiative of the engineers who have constructed the Alpine tunnels; and such a survey of their work as this book presents will be forever a treasure to their successors.—R. W. R.]

DER GRAPHIT. By Ed. Donath. Leipzig, 1904. (Purchase.)

GREAT BRITAIN. MINES AND QUARRIES. General Report with Statistics, 1910, Part IV. London, 1912. (Exchange.)

OTTO HÜBNERS GEOGRAPHISCHE-STATISTISCHE TABELLEN ALLER LÄNDER DER ERDE. 61 Ausgabe, 1912. Frankfurt a/M., 1912. (Purchase.)

AN INTENSIVE STUDY OF THE WATER RESOURCES OF A PART OF OWENS VALLEY, CALIFORNIA. (Water Supply Paper No. 294, U. S. Geological Survey.) Washington, 1912. (Exchange.)

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KONIGLICH PREUSSISCHEN GEOLOGISCHEN LANDESANSTALT ZU BERLIN. Jahrbuch. Band XXIX., teil II., 1908. Berlin, 1912. (Exchange.)

LABORATORY COMPANION TO FATS AND OILS INDUSTRIES. By J. Lewkowitsch. London, 1901. (Purchase.)

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LIABILITY OF THE UNITED STATES FOR USE OF PATENTED INVENTIONS; with special reference to the Act of Congress Entitled "An Act to Provide Additional Protection for Owners of Patents of the United States, and for other Purposes" approved June 25, 1910. By Geo. A. King. Washington, n. d. (Gift of King & King.)

MAP OF CHILE. N. p., n. d. (Purchase.)

MAP OF CUBA. Washington, 1911. (Purchase.)

MAP OF GUATEMALA: Compiled and drawn by M. Hendges. Washington, 1902. (Purchase.)

MAP OF LOCATION OF IRON ORE DEPOSITS, BLAST FURNACES, LIGNITE MINES IN OPERATION AND PRODUCING OIL-FIELDS IN EAST TEXAS, May, 1912. (Exchange.)

MAP OF MEXICO. Compiled and drawn by M. Hendges. Washington, 1900. (Purchase.)

MAPA DE LA REPUBLICA DE BOLIVIA. La Paz, 1901. (Purchase.)

MEXICO. A GENERAL SKETCH. Compiled by the Pan American Union. Washington, 1911. (Purchase.)

MICHIGAN GEOLOGICAL AND BIOLOGICAL SURVEY. Surface Geology and Agricultural Conditions of the Southern Peninsula of Michigan. (Publication 9, Geological Series 7.) Lansing, 1912. (Exchange.)

MINING AND QUARRY INDUSTRY OF NEW YORK STATE. Report of Operations and Production during 1911. Bulletin No. 161, N. Y. State Museum. Albany, 1912. (Exchange.)

NATIONAL EMPLOYMENT EXCHANGE. Annual Report, 3d, 1912. New York, 1912. (Gift of National Employment Exchange.)

NEW ORLEANS. SEWERAGE AND WATER BOARD. Semi-Annual Report, 25th, 1912. New Orleans, 1912. (Gift of Sewerage and Water Board of New Orleans.)

NITRATE DEPOSITS. (Bulletin No. 523, U. S. Geological Survey.) Washington, 1912. (Exchange.)

ALFRED NOBEL DER ERFINDER DES DYNAMITS UND GRÜNDER DER NOBELSTISTUNG. By Rich. Hennig. Stuttgart, n. d. (Purchase.)

NORTH CAROLINA GEOLOGICAL AND ECONOMIC SURVEY. Economic Paper Nos. 27, 28. Raleigh, 1912. (Exchange.)

OIL FINDING, AN INTRODUCTION TO THE GEOLOGICAL STUDY OF PETROLEUM. By E. H. Cunningham Craig. London, 1912. (Purchase.)

OKLAHOMA GEOLOGICAL SURVEY. Bulletin No. 9. Norman, 1912. (Exchange.)

ORGAN DES "VEREIN DER BOHRTECHNIKER." Vols. 1–11. Wien, 1894–1904. (Purchase.)

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PEABODY INSTITUTE. Annual Report, 45th, 1912. Baltimore, 1912. (Exchange.)

PLIOCENE AND PLEISTOCENE FORAMINIFERA FROM SOUTHERN CALIFORNIA. (Bulletin No. 513, U. S. Geological Survey.) Washington, 1912. (Exchange.)

DIE PRAXIS DES EISENHÜTTENCHEMIKERS. By Carl Krug. Berlin, 1912. (Purchase.)

RAILWAY LIBRARY, 1911. Compiled by Slason Thompson. Chicago, 1912. (Gift of Author.)

RESULTS OF SPIRIT LEVELING IN OHIO, 1911. (Bulletin No. 518, U. S. Geological Survey.) Washington, 1912. (Exchange.)

RESULTS OF SPIRIT LEVELING IN PENNSYLVANIA. 1899–1911, inclusive. (Bulletin No. 515, U. S. Geological Survey.) Washington, 1912. (Exchange.)

SICHERHEITSEINRICHTUNGEN IN CHEMISCHEN BETRIEBEN. By Konrad Hartmann. Leipzig, 1911. (Purchase.)

STAFFORDSHIRE IRON AND STEEL INSTITUTE. Proceedings. Vol. XXVII. Stourbridge, 1912. (Exchange.)

STRIKES AND LOCK-OUTS. Memoranda prepared from information in the possession of the Labour Department of the Board of Trade relating to the text and operation of certain laws in the British Dominions and Foreign Countries affecting Strikes, and Lock-outs with especial reference to Public Utility Services. London, 1912. (Purchase.)

STUDY OF A CONTACT METAMORPHIC ORE DEPOSIT. The Dolores Mine, at Matehuala, S. L. P., Mexico. By J. E. Spurr, G. H. Garrey and C. H. Fenner. (Reprinted from *Economic Geology*, Vol. VII., No. 5, Aug., 1912.) (Gift of Bureau of Industrial Research, Washington.

SURFACE WATER SUPPLY OF THE UNITED STATES, 1910, Parts IV., IX., X. (Water Supply Paper Nos. 284, 289, 290, U. S. Geological Survey.) Washington, 1912. (Exchange.)

TASMANIA SECRETARY FOR MINES. Report, 1911. Tasmania, 1912. (Exchange.)

DIE TEKTONIK DES EMSER GANGZUGES NEBST EINER BETRACHTUNG ÜBER ANWENDUNG TEKTONISCHER BEGRIFFE IN BERGBAU UND GEOLOGIE. By Adolf Krümmer. Berlin, 1912. (Gift of Author.)

TREATISE ON GENERAL AND INDUSTRIAL INORGANIC CHEMISTRY. Ed. 3. By Ettore Molinari. Translated by Ernest Feilmann. Philadelphia, 1912. Price, $6 net. (Gift of P. Blakiston's Son & Co.)

[NOTE.—This is a translation by Dr. Ernest Feilmann, of London, of the third and enlarged Italian edition of Prof. Molinari's treatise. It is not surprising that this work has been thus translated; for it is really the most clear, comprehensive and compact modern manual of its class.

Of its four divisions, Part I. is a general introduction, dealing with the fundamental conception and laws of modern chemistry (which, needless to say, include much that used to be called physics). The subjects of matter in the solid, the liquid, and the gaseous state, with consideration of chemical and thermochemical equations, specific heat, the phenomena of dilute solutions, osmotic pressure, surface- and vapor-tension, ions, colloidal solutions, crystallization, "liquid crystals," "phases," radio-activity, etc.

Part II. deals with the chemistry of the non-metals, and Part III. with that of the metals. Under these heads, an immense amount of information is given, with continual reference to industrial applications. Abundant references to authorities, several hundred really helpful illustrations, and an elaborate index, make the volume specially useful to students; and, in turning over the leaves, I have been somewhat surprised to find it highly fascinating to the general reader, as a well-drawn and well-balanced picture of the present state of theoretical and applied chemistry—a quality which is not always prominent in technical treatises prepared for a technical public only. The book contains 704 pages of more than 700 words to the page, and is therefore the equivalent of two volumes of larger type. The typography is beautifully clear, and the translation, which Dr. Feilmann declares to be conscientiously faithful to the Italian text, is singularly free from foreign idioms or awkward periphrases. It reads like an English original—and one of very good style, too.—R. W. R.]

U. S. COAST AND GEODETIC SURVEY. Results of Observations made at the Coast and Geodetic Survey magnetic observatory at Sitka, Alaska, 1909 and 1910. Washington, 1912. (Exchange.)

U. S. GEOLOGICAL SURVEY. Bulletin Nos. 513, 515, 518, 523. Washington, 1912. (Exchange.)

——— Water Supply Paper Nos. 284, 289, 290, 294, 298. Washington, 1912. (Exchange.)

U. S. INTERSTATE COMMERCE COMMISSION. Bulletin of Revenues and Expenses of Steam Roads in the United States, July, 1912. Washington, 1912. (Gift of Interstate Commerce Commission.)

WASHINGTON GEOLOGICAL SURVEY. Bulletin No. 14. Olympia, 1912. (Exchange.)

WATER RESOURCES OF CALIFORNIA. Part I., Stream Measurements in Sacramento River Basin. (Water Supply Paper No. 298, U. S. Geological Survey.) Washington, 1912. (Exchange.)

WESTERN AUSTRALIA. GEOLOGICAL SURVEY. Annual Progress Report, 1911. Perth, 1912. (Exchange.)

GIFT OF HILL PUBLISHING CO.

ABANGAREZ GOLD FIELDS OF COSTA RICA. Annual Report to the Stockholders, 5th, 1911. New York, 1912.

CANADA. CUSTOMS DEPARTMENT. Trade and Navigation unrevised monthly statements of Imports entered for consumption, Aug., 1912. Ottawa, 1912.

CHINO COPPER CO. First quarterly report covering fourth quarter, 1911. New York, 1912.

DAYS OF OLD AND DAYS OF GOLD IN BRITISH COLUMBIA. A few reminiscences of the early gold mining days. N. p., 1912.

GRÈCE. MINISTÈRE DES FINANCES. Bulletin Trimestriel du Commerce Special de la Grece. No. 1, 1912. Athens, 1912.

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QUINCY MINING CO. Report, 1911. New York, 1912.

U. S. TREASURY DEPARTMENT. Comptroller of the Currency. Annual Report, 1911. Washington, 1912.

WEIDIG, M. Radioaktive Wässer in Sachsen. IV. teil. Freiberg, 1912.

WÜST, F. Mitteilungen aus dem Eisenhüttenmännischen Institut der Königl. Techn. Hochschule Aachen. Vierter Band. Halle a/S., 1911.

GIFT OF WEST VIRGINIA MINING ASSOCIATION.

ANNUAL MEETING OF WEST VIRGINIA MINING ASSOCIATION. 1910.

MEETING OF THE EXECUTIVE COMMITTEE OF WEST VIRGINIA MINING ASSOCIATION. Sept. 2, 1908.

EDUCATION OF MINE MANAGERS IN RESCUE WORK, AND THE DANGERS INVOLVED BY UNTRAINED MEN IN ATTEMPTING SUCH WORK. By Wm. Leckie. Oct. 7, 1908.

MINING LEGISLATION IN WEST VIRGINIA. By E. W. Knight. Oct. 7, 1908.

PROFIT AND LOSS ACCOUNT. By Neil Robinson. Dec. 6–8, 1910.

RECENT MINE DISASTERS. Causes and Remedies as far as ascertained. By J. S. Cunningham. Oct. 7, 1908.

STANDARDIZATION OF EXPLOSIVES FOR USE IN MINES. By J. W. Heron. Oct. 7, 1908.

WEST VIRGINIA ON THE BRINK OF A LABOR STRUGGLE. By Neil Robinson. Aug. 28, 1912.

WHY WEST VIRGINIA COAL OPERATORS OPPOSE ORGANIZATION. An Argument.

TRADE CATALOGUES.

GENERAL ELECTRIC Co., Schenectady, N. Y.
  Bulletin No. A 4039. . Direct Current Motor-Starting and Speed-Regulating Rheostats and Panels, 1912. 35 pages.
  Bulletin No. A 4040. Contactors for Industrial Service, 1912. 10 pages.
  Bulletin No. 4993. Type RI., Single-phase Motors, 1912. 15 pages.

A. LESCHEN & SONS ROPE Co., St. Louis, Mo. Leschen's Hercules, Nov., 1912.

MCKIERNAN-TERRY DRILL Co., New York, N. Y. Pile Hammers. 27 pages.

STEPHENS-ADAMSON MFG. Co., Aurora, Ill. The Labor Saver, Nov., 1012.

United Engineering Society Library.

AMERICAN LIBRARY ASSOCIATION. Bulletin, July, 1912. (Papers and Proceedings of the Ottawa Conference.) Chicago, 1912. (Gift of W. P. Cutter.)

GESCHAFTS-BERICHT DER DEUTSCHEN BUCHDRUCKER BERUFS GENOSSENSCHAFT, JAHR, 1911. Frankfurt am Main, 1911. (Gift of Deutschen Buchdrucker Berufs Genossenschaft.)

TRADE CATALOGUES.

BUFFALO WIRE WORKS Co., Buffalo, N. Y. Catalogue No. 5, 1904. 111 pages.

DE WITT WIRE CLOTH Co., New York, N. Y. Catalogue No. 80–909. 148 pages.
  Wire cloth price list and standard gauges adopted March 7, 1899, and March 14, 1900. 40 pages.
  Revised list of wire ropes and wire cords, 1911. 11 pages.

ESTEY WIRE WORKS Co., New York, N. Y. Catalogue and price-list (wire work), No. 20. 63 pages.
  Catalogue No. 19 (wire work). 68 pages.

LIBERTY MANUFACTURING Co., Pittsburg, Pa. Circular of Boiler-Tube Cleaner. 4 pages.

NEW JERSEY WIRE CLOTH Co., Trenton, N. J. Catalogue No. 62, 1911. 45 pages.

PARKER WIRE GOODS Co., Worcester, Mass. Catalogue No. 3, June, 1909. 200 pages.

WRIGHT WIRE Co., Worcester, Mass. General Catalogue of wire, wire fabrics and wire products, 1912. 224 pages.

MEMBERSHIP.

NEW MEMBERS.

The following list comprises the names of those persons elected as members who accepted election during the month of November, 1912:

Members.

Alden, Herbert W., Mech. Engr..............33 Warren Ave. E., Detroit, Mich.
Bull, Robert A., Genl. Supt..........Commonwealth Steel Co., Granite City, Ill.
Brooks, John Y, Mgr., Alabama City Wks., Southern Iron & Steel Co., 2714 Ensley Ave., Ensley, Ala.
Chamberlain, Morrow, Secy. and Treas., Roane Iron Co., Chattanooga, Tenn.
Crandall, Roderic, Geol., Brazilian Geol. Survey, Care W. A. Manning, Palo Alto, Cal.
Emerson, Edward H., Min. Engr.....Cuba Copper Co., Santiago de Cuba, Cuba.
Fyfe, Alexander, Met. Engr.........P. O. Box 400, Haileybury, Ont., Canada.
Goedicke, Carl, Mine Mgr............................P. O. Box 535, San Antonio, Tex.
Goetter, Frederick B., Mill & Cyanide Supt., Mexican Candelaria Co., S. A., San Dimas, Dur., Mexico.
Hendricks, George F., Min. Engr., Nipissing Mining Co., Cobalt, Ont., Canada.
Joralemon, Ira B., Min. Geol., Calumet & Arizona Mining Co., Warren, Ariz.
Loring, Edward A., Min. Engr.......62 London Wall, London, E. C., England.
McKee, Arthur G., Contr. and Cons Engr., 1303 Rockefeller Bldg., Cleveland, Ohio.
Maynard, Edward W., Chem., E. I. du Pont de Nemours Powder Co., Senter, Mich.
Meyers, William R., Min Engr......Cleveland-Cliffs Iron Co., Princeton, Mich.
Notman, Arthur, Min. Geol............................P. O. Box 1062, Bisbee, Ariz.
Reed, Alfred C., Civ. Engr.......................................Felton, Oriente, Cuba.
Rood, Vernon S., Mine. Supt., Utah Apex Mining Co., Bingham Canyon, Utah.
Rowe, Fremont S., Mine Supt............................Hecla Mine, Burke, Idaho.
Sinclair, M. J., Mine Supt...Kellogg, Idaho.
Stillman, Charles A., Secy. and Treas .....Iroquois Iron Co., So. Chicago, Ill.
Tate, E. L., Mine Mgr.................................P. O. Box 384, Spokane, Wash.
Treat, Lloyd B., Min. Engr.....8th Avenida Sur. No. 16, Mexico City, Mexico.
Watanabe, Wataru, Prof. of Min. and Met., Imperial University of Tokyo, 34 Hayashi-cho, Hongo, Tokyo, Japan.
Wilson, Frederick C., Asst. Supt., Spanish-American Iron Co., Felton, Oriente, Cuba.
Yeager, Howard M., Min. Engr.........Continental Coal Corpn., Wallsend, Ky.

Associates.

Elkan, Benno, Mgr. Am. Branch, Beer, Sondheimer & Co., 42 Broadway, New York, N. Y.
Graybill, John H., Electrical Inspector, Jones & Laughlin Steel Co., Pittsburg, Pa.
Holmes, R. G., Salesman, Denver Rock Drill Co., Care United Iron Works, Spokane, Wash.
Johnston, Roderick L., Tank House Foreman, U. S. Metals Refining Co., Chrome, N. J.

CANDIDATES FOR MEMBERSHIP.

The following persons have been proposed during the month of November, 1912, for election as members of the Institute. Their names are published for the information of members and associates, from whom the Committee on Membership earnestly invites confidential communications, favorable or unfavorable, concerning these candidates. A sufficient period (varying in the discretion of the Committee, according to the residence of the candidate) will be allowed for the reception of such communications, before any action upon these names by the Committee. After the lapse of this period, the Committee will recommend action by the Council, which has the power of final election.

Members.

Harry Boyle, Seattle, Wash.
Proposed by A. F. Blair, I. F. Laucks, Chester F. Lee.

John Edward Brantly, University, Ala.
Proposed by Eugene A. Smith, H. B. Litchman, J. M. Shools.

Thomas Ewing Carter, Chattanooga, Tenn.
Proposed by T. Poole Maynard, H. B. Litchman, Eugene A. Smith.

William Henry Casto, Jr., Garfield, Utah.
Proposed by Richard S. McCaffery, Francis Jenkins, C. A. Stewart.

Jesse S. Cheyney, Glamorgan, Wise Co., Va.
Proposed by James Gayley, W. R. Walker.

John Ernest Davis, Villa Nova de Lima, Minas, Brazil.
Proposed by Walter T. Holberton, H. A. Kellar, Franklin Guiterman.

W. Lee Heidenreich, Newhouse, Utah.
Proposed by James F. Kemp, Charles P. Berkey, William Campbell.

Charles M. Heron, Yzabal, Sonora, Mexico.
Proposed by S. W. Mudd, Philip Wiseman, Frank A. Keith.

Charles Ransom Hill, Boston, Mass.
Proposed by Robert H. Richards, Charles E. Locke, H. O. Hofman.

Edwin Alexander McKanna, Oklahoma City, Okla.
Proposed by W. Earl Greenough, James F. McCarthy, William J. Hall.

Angus William Macdonald, Sydney, Nova Scotia.
Proposed by F. E. Lucas, Joseph W. Revere, J. H. Gray.

James F. Magee, Santiago de Cuba, Cuba.
Proposed by James E. Little, R. V. McKay, S. A. Barratt.

Walter Curran Mendenhall, Washington, D. C.
Proposed by W. Lindgren, F. L. Ransome, E. W. Parker.

Burt Hewitt Parsons, Keokuk, Iowa.
Proposed by Hugh L. Cooper, Beverly R. Value, David B. Rushmore, O. B. Perry.

Axel Morton Peterson, Brussels, Belgium.
Proposed by S. H. Ball, Donald Steel, Lucius W. Mayer.

William Robert Phibbs, Mayville, Wis.
Proposed by E. S. O'Connor, B. W. Vallat, A. F. Plock.

Harry M. Pier, Newark, N. J.
Proposed by K. W. McComas, Edward Keller, Hugh D. Pallister.

Yoshio Tsuboi, Akita-ken, Japan.
Proposed by Joseph W. Richards, Willard S. Morse, P. A. Mosman.

Thomas William Van Ess, Unsan, Korea.
Proposed by Alf. Welhaven, J. B. Lower, A. E. Deardorff.

F. R. Wadleigh, Norfolk, Va.
Proposed by W. R. Ingalls, A. T. Shurick, J. T. Beard.

Associates.

Raymond Chase Bergen, Brookline, Mass.
Proposed by Robert H. Richards, H. O. Hofman, Charles E. Locke.

George Davenport Kirkham, Niles, Ohio.
Proposed by A. R. Ledoux, E. Gybbon Spilsbury, Charles F. Rand.

William Albert Rice, Ringwood Manor, N. J.
Proposed by S. H. Hamilton, Guy C. Stoltz, N. V. Hansell.

Robert A. Ricketts, Topeka, Kan.
Proposed by J. deN. Macomb, Jr.

Howard S. Stebbins, Cleveland, Ohio.
Proposed by I. B. Miller, Oscar Textor, C. B. Murray.

Robert Keeler Warner, Syracuse, N. Y.
Proposed by J. F. McClelland, L. W. Bahney, J. D. Irving.

F. S. Wayne, New York, N. Y.
Proposed by H. Foster Bain, Thomas T. Read, E. Gybbon Spilsbury.

Change of Status.

R. Gordon Chaney, Jr., Kenilworth, Utah.

CHANGES OF ADDRESS OF MEMBERS.

The following changes of address of members have been received at the Secretary's office during the month of November, 1912. This list, together with the lists published in *Bulletin* Nos. 63 to 71, March to November, 1912, and the foregoing list of new members, therefore, supplements the annual list of members corrected to Feb. 1, 1912, and brings it up to the date of Dec. 1, 1912.

ADAMS, HUNTINGTON..........Apartado 127, Oaxaca, Oax., Mexico.
ADAMS, WALTER C..........Turkish Bath Hotel, Montreal, Canada.
AMMON, MARK A..........The Willys-Overland Co., Toledo, Ohio.
BARROWS, WALTER A., JR..........Brainerd, Minn.
BIRD, ROBERT MONTGOMERY..........220 Wall St., Bethlehem, Pa.
BRALY, NORMAN B..........822 W. Silver St., Butte, Mont.
BRODERICK, CARLTON T..........162 Congress St., Boston, Mass.
BRUNTON, FREDERICK K., Asst. Supt., British Columbia Copper Co., Greenwood, B. C., Canada.
BURLEY, CURTIS R., Chem..........Hecla Mining Co., Gem, Idaho.
CALDWELL, WILLIAM A., New Ravenswood Mining Co., Ltd., Ravenswood, Queensland, Australia.
COBBE, HERVIC N. G..........86 York St., Westminster, London, S. W., England.
COLE, FRANK L..........Shanghai, China.
COLE, T. F..........1400 Alworth Bldg., Duluth, Minn.
COLVOCORESSES, GEORGE M..........Room 1407, 43 Exchange Pl., New York, N. Y.
COOK, EDWARD H., Mgr. and Owner, Minas de Birimoa, Birimoa, via Canelas, Dur., Mexico.
COSGRO, JOHN P..........Apartado Postal 447, Mexico City, Mexico.
CRANSTON, ROBERT E..........819 Mills Bldg., San Francisco, Cal.
CREMER, FELIX, Cia. de Minas La Blanca y Anexas, S. A., Apartado 142, Pachuca, Hid., Mexico.
DANIEL, JAMES M., JR..........459 Washington Ave., Brooklyn, N. Y.
DEVEREUX, WALTER B., JR..........60 Broadway, New York, N. Y.
EDMONDSON, HORACE W..........6050 Carlton Way, Hollywood, Cal.
EMERY, A. B., Care A. Goerz & Co., Ltd., la de Neuvo Mexico 6, Mexico City, Mexico.
EMMEL, RUDOLPH..........Ludwig, Nev.
EMMENS, NEWTON W..........505 Metropolitan Bldg., Vancouver, B. C., Canada.
EMMONS, N. H..........North Butte Mining Co., Butte, Mont.
FISHER, HOWELL T..........411 Dorchester St., W., Montreal, Canada.
FRASER, ALEXANDER J., Supt..........El Oro Mine, Old Glory, Ariz.
GALLWEY, HARRY A., Genl. Mgr., Butte, Anaconda & Pacific Ry. Co., Anaconda, Mont.
GOODE, E. NOEL, Mt. Morgan Gold Mining Co., Mt. Morgan, Queensland, Australia.
HAMILTON, THOMAS M..........Lewistown, Mont.
HENDERSON, HARRY P..........60 Broadway, New York, N. Y.
HUNT, BERTRAM..........Care E. Hunt & Co., 121 W. George St., Glasgow, Scotland.
IMHOFF, ALEXANDER..........458½ Custer Ave., Los Angeles, Cal.

James, Cyril H., Min. Engr., Bewick, Moreing & Co., 62 London Wall, London, E. C., England.
Jaques, W. H.......................Hotel Puritan, Commonwealth Ave., Boston, Mass.
Kaeding, Henry B.............................4956 Langley St., 2d D., Chicago, Ill.
Kimmel, Harry R....................................532 E. 112th St., Cleveland, Ohio.
Kong, Shun T., Board of Industries and Commerce, Canton, Kwantung, China.
Landers, William H.........................2400 Van Ness Ave., San Francisco, Cal.
Lindgren, Waldemar.......................Mass. Inst. of Technology, Boston, Mass.
Longan, Walker B., Care American Smelting & Refining Co., Charcas, S. L. P., Mexico.
McChrystal, Jackson C.....................723 Kearns Bldg., Salt Lake City, Utah.
McIntyre, John E..P. O. Box 122, Nogales, Ariz.
McKinlay, William B...................................15 Broad St., New York, N. Y.
Malcolmson, Charles T., Prest., Malcolmson Briquet Engrg. Co., Old Colony Bldg., Chicago, Ill.
Manahan, Robert F..............Minas Tecolotes, Santa Barbara, Chih., Mexico.
Menefee, Arthur B., Supt. of Mines, Empire Steel & Iron Co., Oxford, N. J.
Mitinskij, Alexander N...............V. O. 2, line V. 3, St. Petersburg, Russia.
Mitke, Charles A...P. O. Box 1226, Dawson, N. M.
Mooney, James D..................................Care Spokane Hotel, Spokane, Wash.
Montandon, George E...Chuquicamata, Chile, So. Amer.
Montgomery, Ernest A., 1106 Los Angeles Investment Bldg., Los Angeles, Cal.
Newberry, Roger W...............................615 Orange St., New Haven, Conn.
Nichols, Horace G...............................Mawmead, Ashtead, Surrey, England.
Patrick, William B..P. O. Box 32, Millers, Nev.
Pearce, William C. W., Min. and Met. Engr., Mgr., Hadleigh Castle Mine, Lancelot, via Herberton, No. Queensland, Australia.
Pill, John R....................................Corona Coal & Iron Co., Corona, Ala.
Playter, Franklin..205 Hemenway St., Boston, Mass.
Pluemer, A.......................................589 Westminster Ave., Elizabeth, N. J.
Radford, William H., Min. Engr............2360 Broadway, San Francisco, Cal.
Rehfuss, Louis A...Reno, Nev.
Ripley, G. Clinton......................................American Zinc Co., Mascot, Tenn.
Rodgers, Joseph H......................................2357 11th Ave. N., Seattle, Wash.
Rodgers, Myron K......................................2338 10th Ave. N., Seattle, Wash.
Rogers, William B...60 Broadway, New York, N. Y.
Ruggles, Guy H..Clifton, Ariz.
Scott, Walter M. H...........P. O. Box 834, Rio de Janeiro, Brazil, So. Amer.
Short, Frank R.......................... 1057 Monadnock Bldg., San Francisco, Cal.
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Titcomb, Harold A.........................Salisbury House, London, E. C., England.
Turner, R. Chester...Black Hawk, Colo.
Wagner, Percy A..................P. O. Box 427, Salisbury, Rhodesia, So. Africa.
Whitman, Philip R............Supt., U. S. Treasury Mining Co., Chloride, N. M.
Wickes, L. Webster...................................134 Norton Ave., Los Angeles, Cal.
Wilbraham, Arthur G. B., 2 Laurence Pountney Hill, London, E.C., England.
Yates, James...Instructed to hold all mail.

Addresses of Members and Associates Wanted.

Name.	Last Address of Record, from which Mail has been Returned.
Armstrong, Richard E.,	416 E. So. 2d St., Salt Lake City, Utah.
Baxter, Francis K., Jr.,	423 Wells Fargo Bldg., San Francisco, Cal.
Danforth, A. H.,	Cotopaxi. Colo.
David, W. M.,	Care C. G. Hussey & Co., Second Ave., Pittsburg, Pa.
Dyott, George M.,	50 Church St., New York, N. Y.
Eggers, John H., Jr.,	Hughes Creek Mine, Kings River P. O., Cal.
Fergusson, Hugh B.,	216 Loo Bldg., Vancouver, B. C., Canada.
Fitzgerald, Thomas F. M.,	211 Sharon Bldg., Salt Lake City, Utah.
Furness, James W.,	Coffee, Trinity Co., Cal.
Gmelin, Ernest,	Apartado 57, Aguascalientes, Mexico.
Greenley, Louis A.,	Dixie Meadows Mining Co., Prairie City, Ore.

Higgins, Edwin, ... Randsburg, Cal.
Hilson, Cleaveland, ... Livingston, Mont.
Hollis, R. W., ... Silverton, Colo.
Horschitz, Richard J., ... P. O. Box 453, Haileybury, Ont., Canada.
Johnson, Dion L., ... 325 Water St., Pittsburg, Pa.
Lampshire, John O., ... Vulture Mine, Wickenburg, Ariz.
Le Noir, Frank H., ... Box 16, Mt. Bullion, Cal.
McDougall, Wallace D., ... 20 Bedford Place, Russell Sq., London, Eng.
Malins, Francis A., ... Metates Mine, San Marcos, Sin., Mexico.
Moore, Roy W., ... P. O. Box 48, Velasco, Tex.
Munroe, Martin, ... Bengal Coal Co., Murulidih, Mohada, B. N. Ry., Bengal, India.
Nelson, D. W. C., ... Baker City, Ore.
Oldfield, Frank W., ... Mexican Mines Co., Bolaños, Jal., Mexico.
Perks, Harry B., ... 419 Board of Trade Bldg., Portland, Ore.
Peterson, Frank, ... H. W. Hellman Bldg., Los Angeles, Cal.
Rathborne, Merwyn R. W., . . . Amargosa, via Las Vegas, Nev.
Rhew, James W., Cia. Minera y Exploradora de Ventanas, S. A., Ventanas, Dur., Mex.
Rickard, Harold, ... Foley-O'Brien, Ltd., So. Porcupine, Ont., Canada.
Sanders, Bernard H., . . Cartegena Water Wks., Cartegena, Colombia, So. Amer.
Sheldon, Waldo, ... Urique, Chih., Mexico.
Strohecker, John W., ... Rossland, B. C., Canada.
Twynam, Henry, . . . O. K. Copper Mine, Cairns, No. Queensland, Australia.
Vinicombe, Robert E. B., ... Post Restante, Vladikafkas, So. Russia.
Watson, Ralph W., ... Calloo, Utah, Clifton Mail box.
Webster, Erastus H., ... Hotel Cosmopolita, Guadalajara, Jal., Mexico.
White, W. P. . . . . 10 Bernard St., Spokane, Wash.

NECROLOGY.

The deaths of the following members were reported to the Secretary's office during the month of November, 1912:

Date of Election.	Name.	Date of Decease.
1903.	**Carter, Thomas Lane, ...	September —, 1912.
1900.	*Mussen, Horace W., ...	April 18, 1912.

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* Member. ** Life Member.

SUBJECT TO REVISION.

[TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS.]

The Microstructure of Iron and Steel.

BY WILLIAM CAMPBELL, PH.D., SC.D.,* NEW YORK, N. Y.

(Cleveland Meeting, October, 1912.)

THE structure of iron and steel, though the object of so much study and research for the past 25 years, is by no means thoroughly understood.

In the first place, we have three or more distinct iron-carbon thermal diagrams, each with its own supporters, and each representing a theory of the structure and constitution of cast-iron.

In the second place, by reason of the fact that austenite, the solid solution of carbon in iron, decomposes in the solid state into ferrite, or pure iron, and cementite (Fe_3C), and that quenching from above this transition-range, no matter how rapid, cannot entirely prevent the change and thus preserve the austenite by itself at ordinary temperatures, it happens that under the microscope we find, in addition to the ferrite, Fe_3C, and austenite, which the thermal diagram of steel requires, a series of transition-products with distinctive structures, to which have been given the names martensite, troostite, sorbite, etc. Now these transition-products, in addition to the solid solution, austenite, have been the source of much controversy, at times very bitter. One side would insist that, say, troostite was an entity, which the other side would deny, pointing out that it must be a transition-mixture, with a particular structure. That austenite was a solid solution of carbon in iron has been denied by one school; we have been told that $Fe_{24}C$ dissolved both Fe_3C and Fe, and thus gave rise to a whole series of quenching-products—all this in spite of the fact that no matter how the thermal diagrams differ in regard to cast-iron, they all agree in regard to the solid solution, austenite, for $Fe_{24}C$ finds no place on the diagram.

Thus it happens that while the authorities fight both in regard to the iron-carbon diagram and the structures met with in hardened, tempered, and certain alloy-steels, the engineer who would apply metallography to every-day problems is bewildered at the outset by the contradiction of axioms, postulates, and fundamental propositions; and unless he realizes that for him these things are relatively unimportant, he will probably give up metallography in despair. After all, if, under the microscope, we can identify certain structures with certain properties; austenite with strength and toughness and want of magnetism; martensite with extreme hardness, brittleness, and want of ductility; sorbite with strength and ductility, though not hardness; pearlite, the mechanical mixture of Fe and Fe_3C, with ductility, but less strength and hardness than the rest, we have a valuable aid in testing material, even if we cannot define with exactness the ultimate constitution of the different things we recognize under the microscope.

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* Associate Professor of Metallurgy, Columbia University in the City of New York.

In the present paper an attempt has been made to set forth a few of the main features encountered in the microscopic examination of iron and steel, together with some of the points still under discussion.

CLASSIFICATION OF IRON AND STEEL.

I. Wrought-iron:
   1. Puddled iron.
   2. Charcoal-hearth iron (wrought-steel).
   3. Fagoted iron.
   4. Busheled iron.

II. Low-carbon steel:
   1. Pipe-steel, comparison with iron. Welds.
   2. Mild-steel, 0.1 per cent. of C. Segregated bands. MnS, etc.
   3. Structural steel, from 0.2 to 0.3 per cent. of C. Size of grain. Lamination.

III. Medium-carbon steel:
   1. Castings. The diagram. Heat-treatment to refine. MnS.
   2. Axles. "Recrystallization." Heat-treatment.

IV. Rail-steel:
   1. Amount of pearlite. Grain-size. Old rails and new.
   2. Segregation and carbon-content. Flaws.
   3. Tire-steel.

V. High-carbon steel:
    1. 0.75 to 2.0 per cent. of carbon. Normal structure. Effect of reheating.
    2. Graphite precipitation.
    3. Hardening structures. Tempering.

VI. Cement-steel and case-hardening steel.

VII. Alloy-steel:
    1. Pearlitic: Nickel-steel.
    2. Austenitic: Manganese-steel.
    3. Martensitic: Air-hardening steel.

I. *Wrought-Iron.*

Wrought-iron is the product of the treatment of pig-iron in the puddling-furnace or charcoal-hearth, of reheating bundles of scrap-iron and steel (fagoting), of treating miscellaneous scrap-steel in the puddling-furnace (busheling).

There can be distinguished four main types of wrought-iron: 1, puddled iron; 2, charcoal-hearth iron (wrought-steel); 3, fagoted iron; 4, busheled iron.

1. *Puddled Iron* contains (unlike steel, etc.) no manganese, high phosphorus (which is mainly in the slag), as compared with charcoal-hearth iron or ingot-iron; and under the microscope is composed of polygonal grains of ferrite, through which are distributed threads or elongated flakes of slag, Fig. 1. When one considers the process of manufacture, the cause of this structure becomes evident. The metal comes to nature as solid grains and dendrites of pure iron, which are worked up into a ball, and withdrawn from the furnace dripping with molten slag. Next the greater part of the slag is squeezed out under the hammer, or in the squeezer, while a further slight amount is extruded in the puddle-rolls. The product is a muck-bar of coarse grain and coarse slag-fibers. This is cut up, piled, reheated to welding-heat, and rolled out once more. The resulting metal has proportionately finer grains and finer slag-fibers. A second piling, reheating, and rolling will produce a still finer slag-fiber. In addition, a certain amount of oxide also is introduced as fibers and flakes with each piling and reheating.

An examination of the slag of wrought-iron shows that in the case of rounds and squares the form is that of fibers, while with flats and sheets it is necessarily that of flakes and elongated plates. The slag is composite and consists of a dark ground-mass (probably silicate) full of dendrites and round globules of a lighter material, showing a bluish tinge by vertical light (probably oxide). The ferrite matrix is composed of polygonal grains, Fig. 13, the size of which depends on the time and temperature in the reheating-furnace and the amount of reduction of the metal with rolls, presuming that the finishing-temperature is above the critical-point, Ar_3, or, say, 900° C. If, on the other hand, the metal is finished below this point, the grains of ferrite are more or less elongated in consequence. As three different types, we might compare the structure of a piece of large shafting or an old wrought-iron rail with that of a piece of pipe-skelp, and a piece of cold-rolled iron.

2. *Charcoal-Hearth Iron* (*Wrought-Steel*) contains no manganese (being thus distinguished from steel); the phosphorus-content is much lower than that of puddled iron, and the presence of slag distinguishes it from ingot-iron.

In structure, charcoal-iron is very similar to puddled iron, but, owing to the fact that it comes in contact with the glowing bed of charcoal during manufacture, parts of the bloom become carburized, and these appear in the finished sheet or bar as bands of ferrite, containing more or less pearlite. The center of the band contains most pearlite, which gradually fades away towards the outside and passes into pure ferrite. This is typically shown in Fig. 2. The slag has the same characteristics as in puddled iron.

Dannemora iron is converted into blister-steel by cementation, and may be used by remelting in crucibles for the manufacture of crucible-steel. Hence the presence of considerable quantities of carbon is not objectionable. On cutting a cross-section through certain brands, the banded appearance shows very markedly to the eye. Under the microscope we find alternating bands of ferrite and slag, ferrite and pearlite, and even pearlite alone, the one band passing imperceptibly into the next. Such material has been called wrought-steel, when capable of hardening.

3. *Fagoted Iron*, as commonly made, is a mixture of old

wrought-iron and scrap-steel, preferably low in carbon. When the scrap is small, a "box-pile" is made, with base and sides composed of sheet wrought-iron, and the interior is filled with a mixture of iron and scrap-steel. The whole is heated to welding-heat and then rolled out, as in the manufacture of puddled iron. Hence this material usually contains an appreciable amount of manganese. In structure it resembles charcoal-hearth iron, for it contains bands of ferrite and pearlite; but these bands, due to the scrap-steel, do not pass imperceptibly into the ferrite (and slag) areas as in charcoal-hearth iron. There is generally a sharp line of division between the two, with a more or less continuous band of slag and oxide fibers, which is seldom the case in charcoal-hearth iron. Fig. 14 is a cross-section of such material.

4. *Busheled Iron.*—By busheling scrap is meant the charging of the puddling-furnace with baskets of small steel scrap, bringing them to incipient fusion, balling the mass, and squeezing out and rolling as in the true puddling process. This produces a bar or sheet composed of particles of steel compactly welded together, with intermingled slag, closely resembling puddled iron, but containing much more manganese (and generally less phosphorus). The structure of this material is extremely hard to distinguish from puddled iron on the one hand and charcoal-hearth iron on the other, because the distinct lines of separation between iron and steel as formed in fagoted material are missing.

On the other hand, if the steel scrap is actually melted in the puddling-furnace, there is nothing in etching to distinguish the structure of the finished product from wrought-iron.

II. *Low-Carbon Steel.*

For the sake of convenience there is included in this group material up to 0.3 per cent. of carbon, such as is used for structural purposes.

1. *Pipe-Steel and Skelp.*—The chief question in microscopic examination is to decide whether a pipe is composed of puddled iron, charcoal-hearth iron, or steel. The following are analyses of these three materials, together with ingot-iron for comparison:

	Mn.	P.	S.	Slag.	Si.	C.
	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.
Puddled iron......	Trace — 0.2	0.12	0.02	1.0	0.2	0.03
Charcoal-iron......	Trace.	0.03	0.007	0.75	0.04	0 — 0.09
Steel	0.35 — 0.45	0.1	0.04	0.1	0.04	0.07 — 0.09
Ingot-iron...........	0.02	0.005	0.015	......	0.005	0.02

The carbon averages 0.03 per cent. in all but steel. Of course, there are many variations from the above, and many cases cannot be decided by analysis or by the microscope alone. We have already discussed the difference between puddled and charcoal-hearth iron. Steel is composed of ferrite with from 8 to 10 per cent. of pearlite, usually occurring as small patches between the grains of the ferrite. In addition, there are small elongated globules of dove-colored manganese sulphide and occasionally thinner black threads of slag or oxide. The structure of both the ferrite and the pearlite differs in skelp, sheet, small and large pipe. In the sheet the ferrite is usually in polygonal grains of small size, in small pipe the grains are somewhat larger, but still more or less polygonal, while in large pipe this feature is missing as a rule, and the ferrite grains are more or less mixed up with the pearlite scattered in and between them. This is owing to the temperature and the time given in the heating-furnace for welding. Again, in sheet and small pipe the pearlite often shows marked segregation, the ferrite and the pearlite having coalesced with the main mass of the ferrite, while the cementite forms small irregular patches. The same is very common in low-carbon steel rounds of about 0.1 per cent. of C. Light etching with picric acid shows up the pearlite, while prolonged etching will show up the grain of the ferrite also. Picric acid seems preferable to nitric or sulphuric acid, because it yields sharper boundaries to the ferrite grains and does not eat into the material around slag, MnS, or pearlite.

Macroscopic etching is commonly used to distinguish between steel and wrought-iron pipe. After the section has been filed smooth and rubbed down with Nos. 0 and 00 emery-paper, it is deeply etched with nitric, sulphuric, or other etching-medium. The steel shows up plain, while the wrought-iron shows a concentric structure, due to the inclosed slag.

Lap-welds and butt-welds can usually be distinguished in the etched specimen by the eye. Under the microscope the lap-weld is seen to form an obtuse angle with the lamination of the steel or wrought-iron and can be traced by entrained particles of oxide or cinder. The butt-weld can be traced much more readily; for at the outside and the inside of the pipe there is usually a slight V-shaped depression, the bottom filled with oxide, which indicates the direction of the weld. In the metal between these two points a few globules of oxide indicate the weld, which can be recognized also (1) in steel because the pearlite laminations are not continuous; (2) in wrought-iron because the slag globules and threads are discontinuous and often bent over by the force of the weld.

The structure of ingot-iron consists of polygonal grains of ferrite, as in Fig. 13.

Mild-Steel.—Steel with about 0.1 per cent. of C is widely used for light shafting, rods, bands, rivets, spikes, screws, and machine-work.

The analyses vary, depending on whether it is Bessemer or basic open-hearth.

Under the microscope we find it composed of ferrite, with about 10 per cent. of pearlite. The grain-size depends chiefly on the cross-section, or, rather, on the amount of reduction in rolling from the last heating. As in the case of pipe-steel, so here we often find the pearlite segregated into little globules, or flakes of cementite. In a longitudinal section the pearlite generally shows some lamination, which is also seen by the way the MnS (and slag) is elongated. Fig. 3 is a longitudinal section of a 2.5-in. round. The small black patches are pearlite, the long, thin streak across the center is slag. Fig. 4 is a section of a 3.5-in round, and the dark etching band across the center is a "ghost," and owes its presence to segregation of P, MnS, etc., which produces "streaky" steel. In screw-stock the MnS is very marked in the longitudinal sections.

3. *Structural Steel.*—As the carbon is increased from 0.2 to 0.3 per cent. there is a proportionate increase in the pearlite, which is more commonly laminated with the ferrite. Streaks of MnS and occasionally of slag are seen, usually imbedded in bands of ferrite. As before, the slag is black and in much thinner threads than the MnS, because its freezing-point is lower and the plastic range longer, and therefore it tends to become

more strung out in the rolling. The rule generally holds good that the thicker the section the coarser the grain, and the less pronounced the lamination. Fig. 7 is a section of a 2- by 0.25-in. piece, showing the lamination.

III. *Medium-Carbon Steel.*

1. *Castings.*—With regard to the structure of steel with from 0.3 to 0.4 per cent. of carbon which has been rolled, the above remarks hold good. The structure of castings, however, is quite different. When a casting freezes, dendrites of the solid solution austenite tend to grow perpendicular to the cooling-surface, and we get the characteristic "pine-tree" crystallization on the outside. The interior is made up of more or less granular crystals. On cooling down there is no change until the temperature reaches the upper critical point, Ar_{2-3}, or the line *GOS* of the iron-carbon diagram,[1] Fig. 25, when ferrite commences to separate out around and in the grain of the austenite, giving the characteristic austenitic structure resembling the Widmanstätten figures of meteorites, or, if the carbon be high, the feathery ferrite. On reaching the lower critical point, Ar_1, or *PSK* of the diagram, from 700° to 660° C. (depending on the manganese), the remaining austenite, which now contains approximately 0.9 per cent. of carbon, splits up into the eutectoid pearlite, the mechanical mixture of ferrite and cementite. The structure of this pearlite and perhaps its content of carbon depend largely on the rate at which the steel passes through this critical temperature. Rapid cooling in air, for instance, yields very fine pearlite, while very slow cooling gives us the coarsely laminated variety whose pearly appearance is marked to the naked eye. Fig. 9 shows a steel with C, 0.35, and Mn, 0.66 per cent., as cast.

Such coarsely crystalline material is brittle, and must be annealed, or, more properly speaking, refined. This is done by reheating to above the upper critical point, Ac_{2-3}; that is, just above *GOS*. On passing the line *PSK*, or Ac_1, the pearlite changes over into austenite, and with increase in temperature the solid solution austenite dissolves more and more of the ferrite until, on the Ac_{2-3} change being completed, we have

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[1] See Heyn. Report on Progress of Metallography, *Proceedings of the International Association for Testing Materials*, vol. ii., No. 11 (1912).

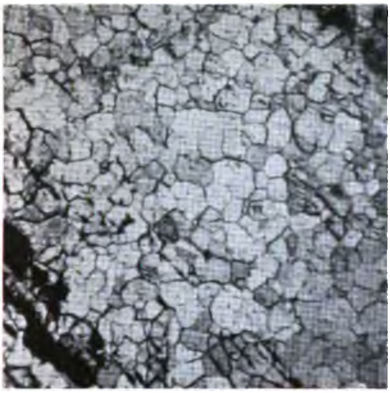

FIG. 1.—WROUGHT-IRON. MAGNIFIED 50 DIAMETERS.

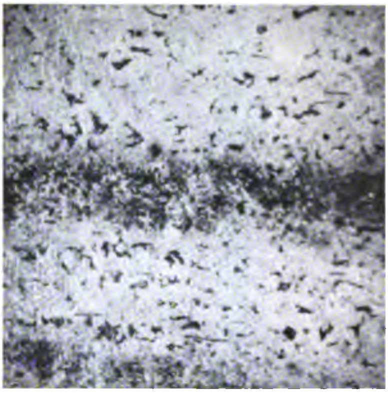

FIG. 4.—LOW-CARBON STEEL. "GHOST." MAGNIFIED 50 DIAMETERS.

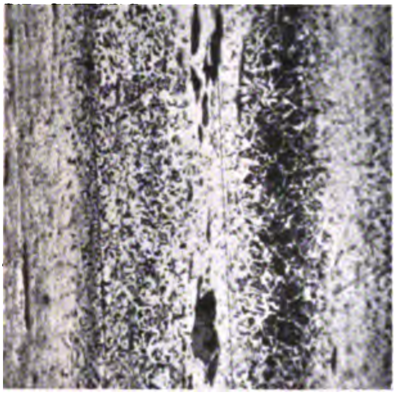

FIG. 2.—CHARCOAL-HEARTH IRON. MAGNIFIED 50 DIAMETERS.

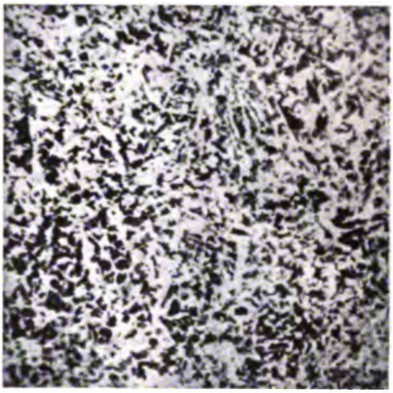

FIG. 5.—REFINED STEEL CASTING. MAGNIFIED 50 DIAMETERS.

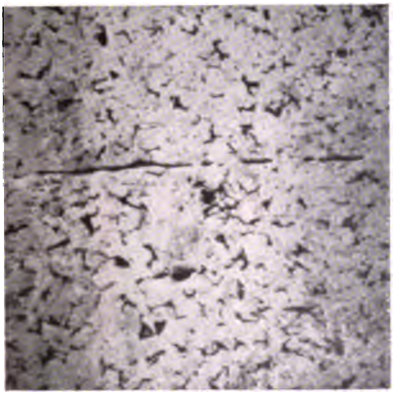

FIG. 3.—LOW-CARBON STEEL. MAGNIFIED 50 DIAMETERS.

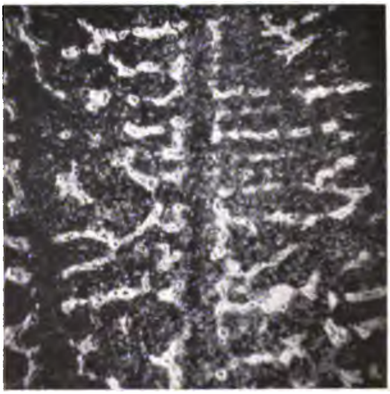

FIG. 6.—MnS., ETC., IN REFINED CASTING. MAGNIFIED 30 DIAMETERS.

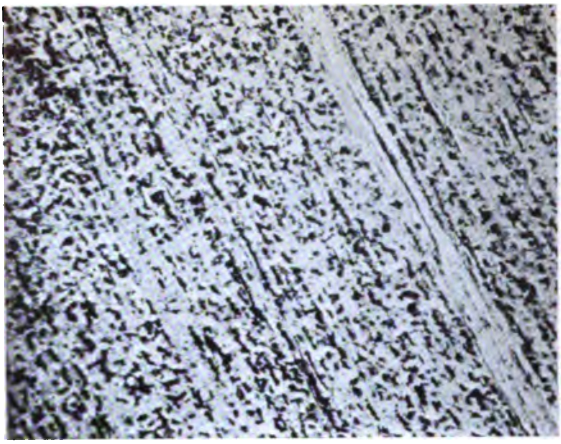

FIG. 7.—STRUCTURAL STEEL. MAGNIFIED 50 DIAMETERS.

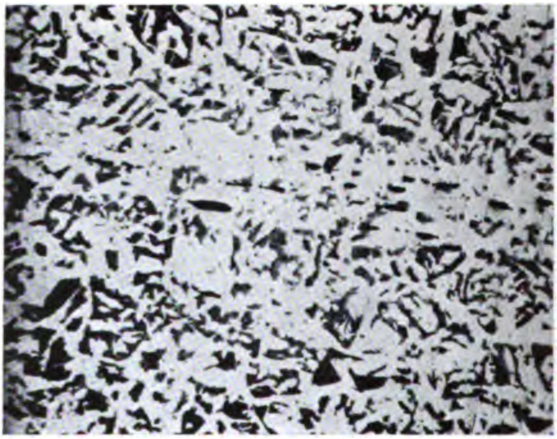

FIG. 8.—3.5-IN. ROUND. MAGNIFIED 50 DIAMETERS.

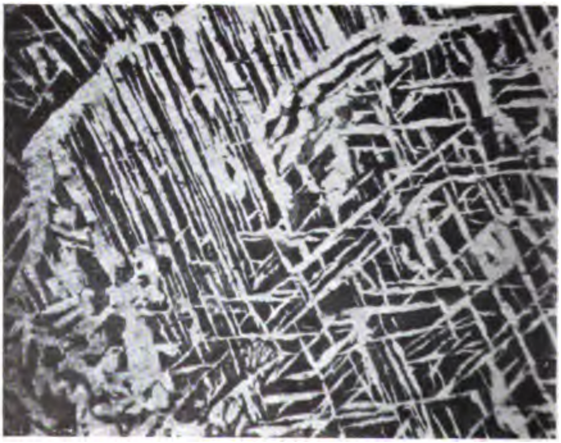

FIG. 9.—STEEL AS CAST. MAGNIFIED 50 DIAMETERS.

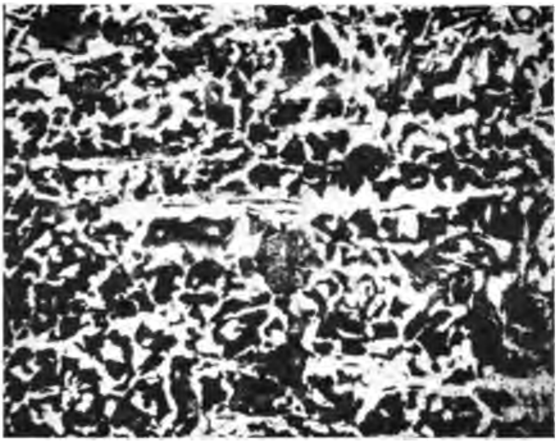

FIG. 10.—3.5-IN. SHAFT. MAGNIFIED 50 DIAMETERS.

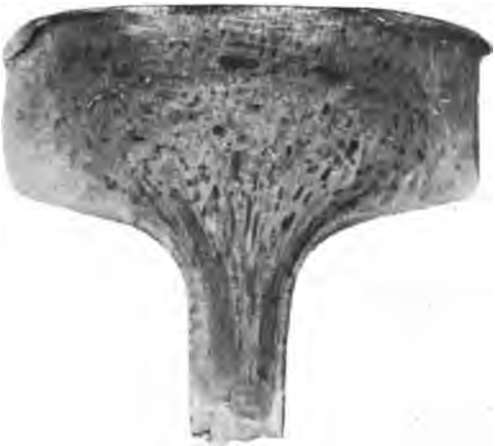

FIG. 11.—RAIL-HEAD. ETCHED.

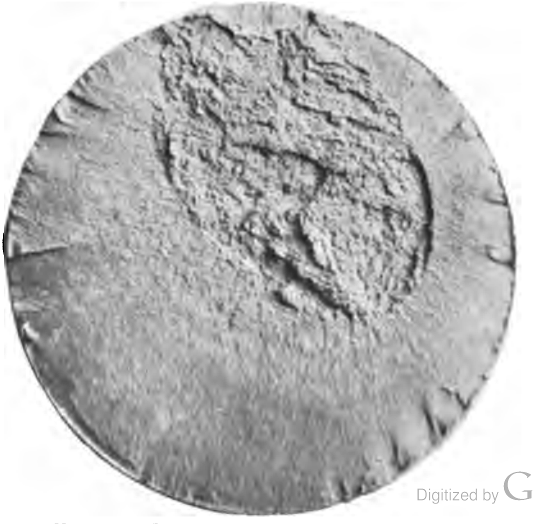

FIG. 12.—BROKEN SHAFT. "FATIGUE."

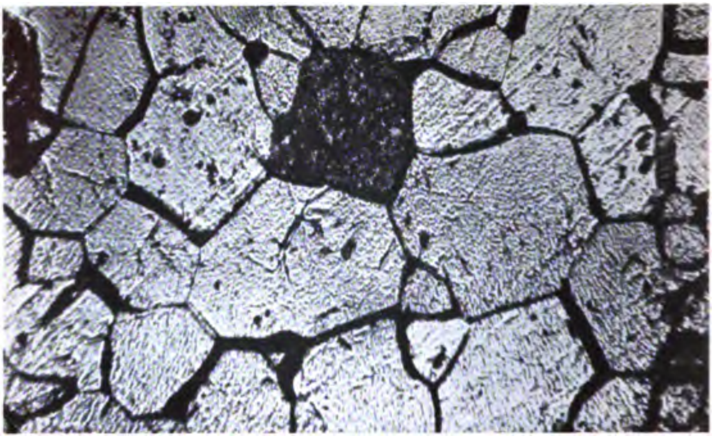

FIG. 13.—WROUGHT-IRON. MAGNIFIED 500 DIAMETERS.

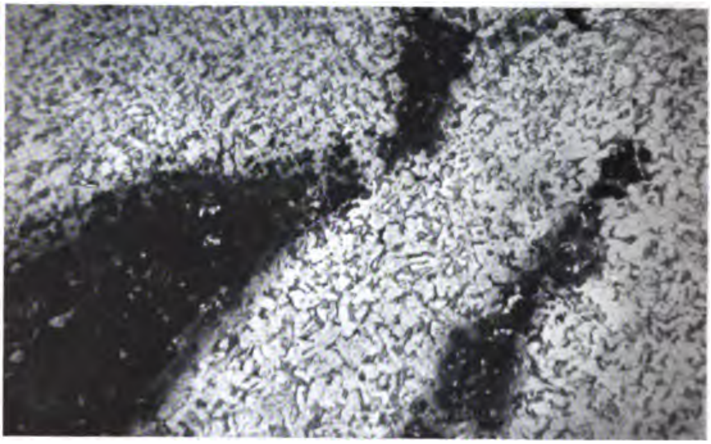

FIG. 14.—WROUGHT-IRON MADE FROM SCRAP. MAGNIFIED 70 DIAMETERS.

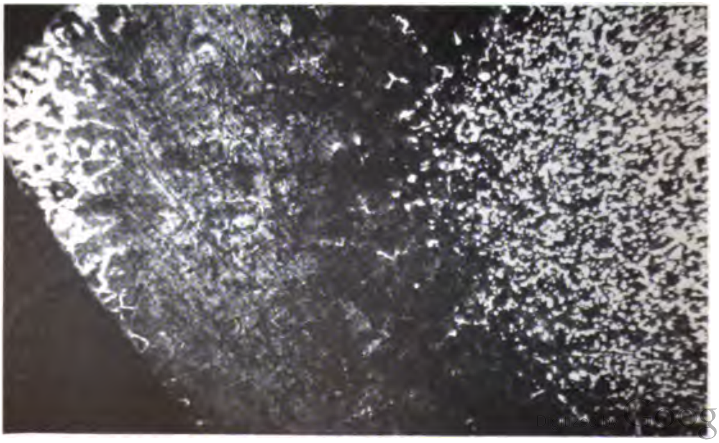

FIG. 15.—2 PER CENT. C. HEATED TO 1,070° C. MAGNIFIED 40 DIAMETERS.

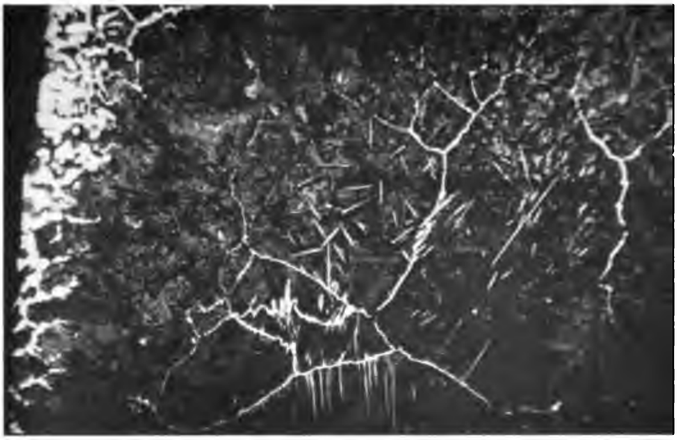

FIG. 16.—2 PER CENT. C. HEATED TO 1,200° C. MAGNIFIED 40 DIAMETERS.

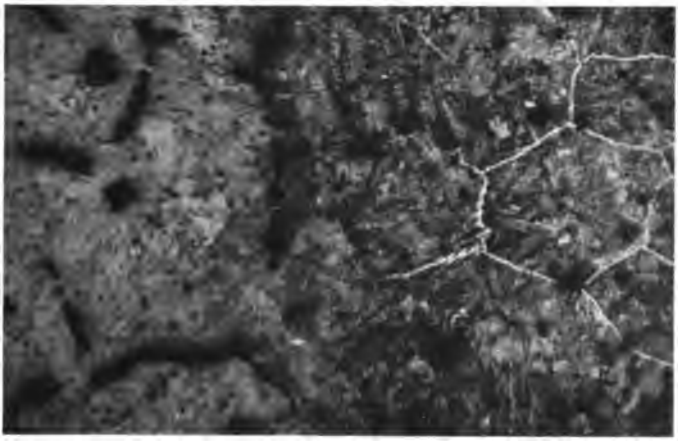

FIG. 17.—GRAPHITE AND CEMENTITE. MAGNIFIED 60 DIAMETERS.

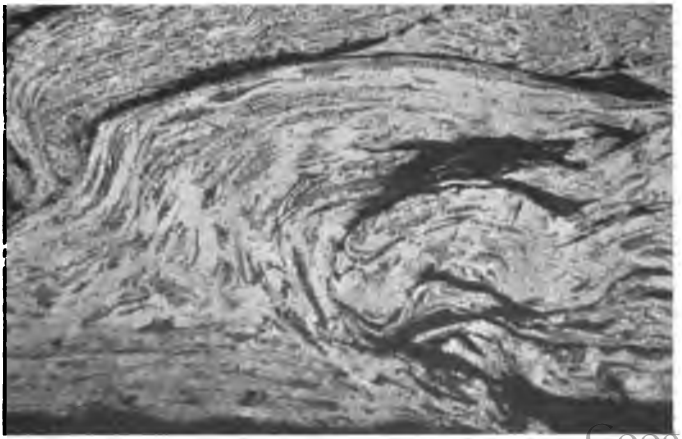

FIG. 18.—TIRE, SHELLING OUT. MAGNIFIED 70 DIAMETERS.

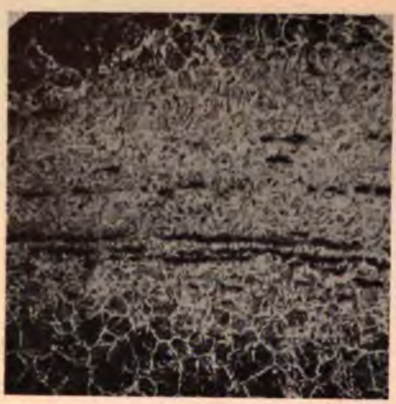

FIG. 19.—SLAG IN RAIL-STEEL. MAGNIFIED 50 DIAMETERS.

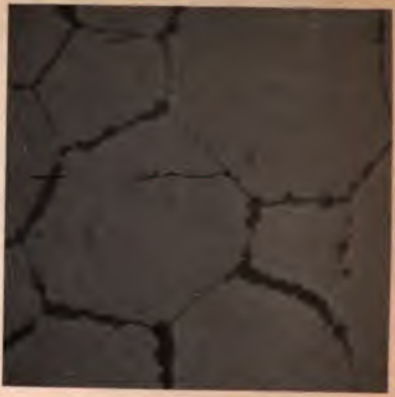

FIG. 22.—AUSTENITE AND TROOSTITE. MAGNIFIED 50 DIAMETERS.

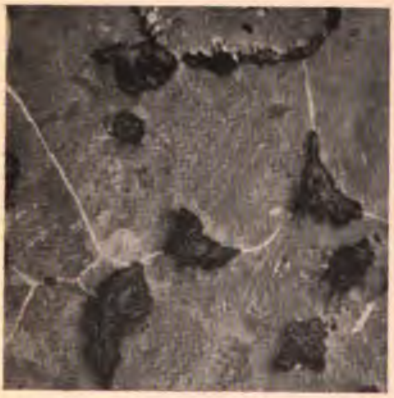

FIG. 20.—TIRE-STEEL, BURNED. MAGNIFIED 60 DIAMETERS.

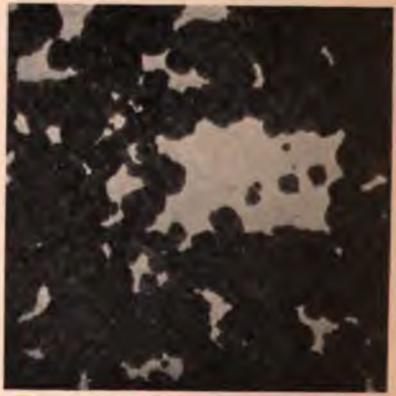

FIG. 23.—TROOSTITE AND AUSTENITE. MAGNIFIED 50 DIAMETERS.

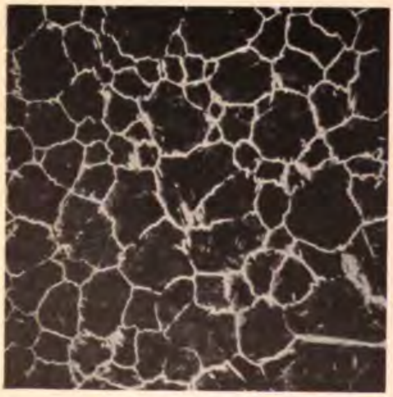

FIG. 21.—CEMENTITE AND PEARLITE. MAGNIFIED 60 DIAMETERS.

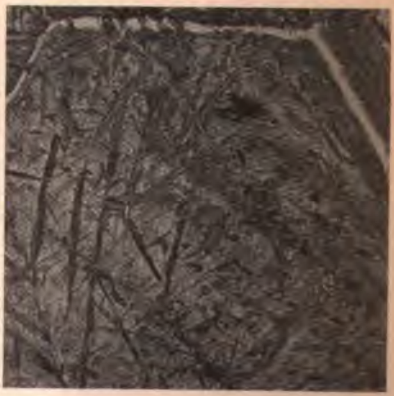

FIG. 24.—CEMENTITE AND MARTENSITE. MAGNIFIED 250 DIAMETERS.

nothing but fine-grained austenite. On cooling down again, the ferrite separates out as before; but now it is confined to very small grains of austenite, and hence the resulting structure is fine. Fig. 5 shows the same steel heated to 850° C. and thus refined.

It is not uncommon to find that such a treatment has not completely refined a specimen. The reason lies in the presence of globules and films of MnS or oxides, which often assume a dendritic form, due to their being deposited in the dendrites of the original casting. Heating to above $Ac_{2\text{-}3}$ truly refines this steel, but on cooling down again the ferrite tends to precipitate on these films and globules of MnS, and lines of weakness result. However, the heat-treatment has materially improved such material, as the tensile tests will show. Fig. 6 is a piece of steel with C, 0.5, and Mn, 0.68 per cent., after refining, and shows the dendritic structure due to manganese sulphide, etc.

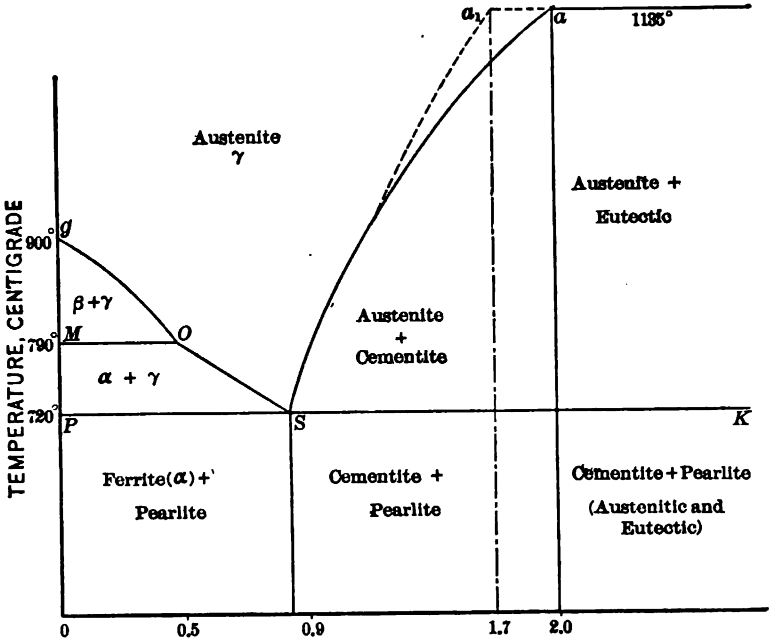

FIG. 25.—AUSTENITE DIAGRAM.

2. *Axle-Steel.*—The structure of axle-steel is similar to that of structural steel, allowing for the increase in pearlite with increase in carbon. Grain-size increases with section. A great many cases of failure have been ascribed to fatigue or recrystallization due to vibration. We see a fracture the outer part of which is very fine and the inner core comparatively coarsely crystalline, as in Fig. 12. On cutting a section from center to outside, the grain-size is found to be practically the same. Of course, there has been no growth of grain. Under repeated strain, a tiny crack starts at the outside and slowly works its way round the section, becoming deeper and deeper. As it progresses, the two faces pound on each other and tend to make the surface smooth. In time the cross-section of the solid metal becomes so small that it can no longer bear the strain, and rupture takes place, giving a normal crystalline fracture. Fig. 8 is a longitudinal section of a 3.5-in. shafting which broke with so-called recrystallization. The structure is the same from outside to center. Through the center of the photograph is a band comparatively rich in ferrite, with characteristic elongated globules of MnS running through it.

Fig. 10 shows the structure of a 3.5-in. shaft which also broke. The carbon-content is much higher, but here again is MnS set in a strip of ferrite.

Thus there are two kinds of fracture: the inside, or normal, bearing a definite relation to the grain-size, and the outside, or abnormal, having no relation whatever to the internal structure of the piece. It is to be noted, however, that such broken material often shows a coarse grain, or much segregation with MnS and associated high-phosphide areas, or both; and it has been proved that reheating to the refining-point and cooling in air, or, better still, quenching and again reheating to from 500° to 600° C., gives material which withstands "fatigue" extremely well.

Axle-steel may therefore be divided into three main groups: forged, "annealed," and "tempered."

The forged material stands up well in use if in perfect alignment, that is to say, if it receives a minimum amount of vibration, or if it is not subject to "fatigue." If the axle or shaft is not properly set, fatigue sets in. It is found that the coarser the grain the more liable are slip-lines to occur, and, therefore, fatigue comes sooner, with a tendency to the progressive fracture we have already described.

With the "annealed," or, more correctly speaking, refined, a fine grain is less likely to produce slip-lines, and the material is therefore less liable to fatigue, other things being equal.

The last group, which has been classed as "fibered" material, especially in nickel-steel, withstands strain and vibration best. If we heat to the refining-temperature, Ac_{2-3}, and quench, we get material made up of extremely fine austenite, etc. On reheating to from 500° to 600° C. there is a great change within this tiny grain itself. If we fracture a longitudinal section it has a distinctly fibrous appearance. It is much finer than the structure found in wrought-iron. This fibrous appearance is due entirely to the method in which the individual grains elongate and break; it is accentuated by traces of MnS, etc., which have been elongated in the direction of rolling.

This material has about the same size of grain as that which has been refined, but the interior of the grain is quite different in the latter. The pearlite is comparatively coarse and, under high power, is distinctly pearly, that is to say, the ferrite and cementite can be resolved. The tempered material is pearlite, but in a much finer condition. It is almost sorbitic; for it is the product of the change from austenite through martenite with troosite into sorbite and pearlite, which takes place on reheating. Such material has been called "anti-fatigue;" and nickel-steel, giving the maximum amount of fibrous structure, is ideal for such material.

IV. *Rail-Steel.*

With an increase in carbon to from 0.4 to 0.6 per cent. there is a further increase in pearlite. In estimating the amount of carbon by the microscrope, both the amount of manganese (up to 1 per cent.) and the comparatively rapid cooling in air have to be taken into account. Thus, a low-carbon steel with about 10 per cent. of pearlite will contain about 0.09 per cent. of C, the factor being 0.9. With from 25 to 30 per cent. of pearlite the factor is about 0.8 and there is, say, from 0.20 to 0.25 per cent. of C. At 50 per cent. of pearlite, multiplying by 0.75 gives about 0.37 per cent. of C. With 75 per cent. of pearlite and a factor of 0.7 there is about 0.42 per cent. of C; while at 0.6 of C there is only a trace of ferrite left. Of course, with slowly-cooled material the factor remains about 0.9. The grain-size depends mainly on the section, that is, the amount of reduction in the rolls, rather than the finishing-temperature.

In examining a rail-section, it will be found that the grain-size varies considerably. The center of the head will be comparatively coarse, the web much finer, while the edge of the flange is comparatively fine and often shows marked lamination of the ferrite (Fig. 10). Macroscopic etching of a cross-section will show up any segregation that may be present. Fig. 11 shows a section of a rail-head which failed in service. It has been etched with iodine solution and the dark spots show segregation of C and also of MnS. Under the microscope, these dark areas are quite different from the normal section. A longitudinal section shows this well: the steel appears quite streaky, each streak running many inches, sometimes feet. MnS globules in elongated form, also occasionally thin streaks of slag or oxide, usually form the core of such a streak.

The question is frequently raised, whether the old rails were not better than those made to-day; whether our metallurgical progress has not sacrificed quality for tonnage. Undoubtedly, rail-failures are far too frequent; and, in looking for the cause, a careful study was made of a number of old rails, as well as new, to see if the microstructure would tell anything. The grain-size showed but little difference, the size of section having apparently controlled the grain. Macroscopic etching, while showing some of the old rails to be absolutely free from segregation, detected extreme segregation in others—quite as bad as that found in present-day rails. Under the microscope, however, there is a marked difference, due to the carbon-content; in other words, the old rails all had a large excess of ferrite. Their specifications called for from 0.35 to 0.45 per cent. of C. This gave a large excess of ferrite, which shows up distinctly laminated in longitudinal sections (Figs. 8 and 10). Locally-segregated areas, as in Fig. 11, showed an increase in the amount of pearlite, but in the worst cases there was still a comparatively large excess of ferrite, which seemed to minimize the effects of MnS, "ghosts," etc. The modern rails, with carbon from 0.5 to 0.65 and even higher, showed the same segregated areas as in Fig. 11. Under the microscope, however, these areas no longer show the excess of ferrite, and the same amounts of MnS, oxide or slag, "ghosts," etc., are now the causes of extreme brittleness.

In regard to piping and consequent segregation, the 0.35 to 0.45 per cent. carbon rail would show at base of head and top of web a maximum segregation of, say, 0.65 per cent. of C. The modern, heavier rail, with from 0.5 to 0.65 per cent. of C and more than 1 per cent. of Mn, frequently shows segregation up to far above 1 per cent. of C, and under the microscope are seen envelopes and plates of cementite, Fe_3C, in and around the pearlite grains, causing extreme hardness and brittleness. Fig. 16 closely resembles such extreme segregation, the thin white envelopes and needles being cementite set in a ground-mass of pearlite. Split heads in rails are mainly due to segregation, occurring in rails from the top of the ingot chiefly. Dudley has pointed out the wedge-action of the hard spots, MnS, slag, etc., which finally causes the head to split vertically. Wickhorst has pointed out the flow of metal on the surface of the head, as shown in Fig. 11, which soon tends to tear apart the head, and in segregated rails a vertical crack occurs in the harder and more brittle interior.

Surface-cracks, checks, and seams in the head are generally found by the inspector. They are filled with oxide, from a mere film to a thick mass, and this oxide is surrounded by an envelope of ferrite, due to decarburization. A crack which has occurred in service may also be filled with oxide, but in this case signs of decarburization are missing, and there is no relation between the grains and the fissure filled with oxide, or, rather, rust.

Basal seams are not so easy to see on inspection, but are readily revealed by macroscopic etching. Their chief position is under the web, and they are found in sections showing slag or oxide and marked decarburization. Such a seam is shown in Fig. 19, a section parallel to the rolling. The large amount of ferrite shows abnormal decarburization. Now this is the point which receives the most strain when a rail is loose, and, as a line of weakness, it rapidly starts to open up and the flange ultimately breaks.

The best method of examination for rails is to etch a cross-section macroscopically and then make longitudinal sections through the dark or segregated areas. It has been found by several that MnS, alone is not nearly so indicative of brittleness as when accompanied by the thin threads of slag, which generally means the proximity of segregation and the piped area of the ingot.

Tire-Steel.—As usually made, a steel tire shows a fairly-coarse grain, approximately of the size of those in Fig. 22, with thin envelopes of ferrite round the pearlite in steel with 0.6 per cent. of C; while with 0.75 per cent. of C and 0.7 per cent. of Mn the ferrite envelopes disappear. Properly-refined steel shows a very fine grain, even smaller than that in Fig. 21. A Krupp tire generally has a grain of half that size. Some of the Schoen tires showed an extremely fine grain with a sorbitic appearance, closely resembling material quenched from the critical point and reheated to, say, 500° C.

Two kinds of failures have been examined, breakage and "shelling out." In the former, the steel seemed perfectly good, grain-size normal, no marked segregation; and therefore it was concluded that here was another case of "fatigue," that the steel had been subjected to strain at this particular point sufficient to produce progressive fracture. As in the case of axles, so here no trace of any "recrystallization" was met with.

"Shelling out" seems to be caused in three ways: (1) surface-flaws with oxide, similar to Fig. 19; (2) a decarburized surface, which is soft and soon flows, develops cracks which fill with rust until pieces flake off, as shown in Fig. 18, which is a section of a shelled-out tire at the surface; (3) cracks filled with oxide (rust). The first two occur in the process of manufacture; the last occurs in service with apparently good steel. A section will show a certain amount of flow of metal, with cracks set at a slight angle to the surface. These are filled with from films to thick masses of oxide—not slag—due to rusting. There is no decarburization around them, as is usual in the case of slag. It would seem that their cause is similar to that assigned by Wickhorst to certain split-heads in rails. Burning of the steel has often been mentioned as the cause of this kind of failure. A piece of tire was burnt purposely and its structure is shown in Fig. 20. The thin white envelopes to the grains are ferrite, the black masses are oxide. Such a structure has not been met with in tire failures, hence it seem reasonable to assume that burning is not the cause of "shelling out."

V. *High-Carbon Steel.*

As already remarked, steel with C, 0.55, and Mn, 1 per cent., as rolled, consists of pearlite. $Ac_{1,2,3}$ occurs at 710° to 715° C.; $Ar_{1,2,3}$ at 670° to 655° C.

Similarly, steel with C, 0.9, and Mn, 0.2 per cent. (crucible) is all pearlite, and $Ac_{1,2,3}$ occurs at 730° to 735° C., and $Ar_{1,2,3}$ at 710° to 700° C. The differences in temperature are evidently due to the manganese.

Slow cooling past $Ar_{1,2,3}$ produces well-laminated pearlite; while rapid cooling of small sections, as in air, yields a sorbitic structure.

Overheating occurs at a comparatively low temperature. A bar $\frac{5}{16}$ in. sq., containing C, 0.7, and Mn, 0.068 per cent., was overheated at 800° C., while one containing C, 1.04, and Mn, 0.12 per cent., was overheated at 855° C.

At 1 per cent. of C, are found traces of cementite as thin envelopes to the grains. These envelopes increase with the carbon-content until at 2 per cent. of C they are extremely coarse Fig. 21 shows such cementite envelopes in a steel with 1.72 per cent. of C.

The boundary-line between cast-iron and steel is unsettled. According to the generally-accepted definition, steel can be rolled out, but cast-iron cannot. From the iron-carbon diagram, the boundary-line occurs at 2 per cent. of C (or 1.7 per cent. according to Ruff), where the eutectic of austenite and cementite makes its appearance. Now, an ingot with a small amount of eutectic can be rolled out, for this cementite is rolled out and broken up just as slag is drawn out in cold-worked wrought-iron. Material with as much as 2.4 per cent. of C has been rolled. Hence, if the presence of the eutectic denotes cast-iron, cast-irons very low in carbon can be rolled and are malleable.

The heat-treatment of high-carbon steels is not simple. With material of not more than 0.9 per cent. of C, heating to just above the line *GOS* causes refining; above that temperature, the higher the temperature of the heating, the coarser the grain. In steels with over 0.9 per cent. of C, the diagram would seem to require that they be heated to just above the line *Sa* to dissolve the cementite envelopes in the austenite, and therefore to refine. In practice, heating to just above the line *PSK* refines the steel (at all events, in small sections), because, on heating, the cementite tends to draw itself up into globules and refines itself.

With high carbon (1.6 per cent. and over), heating to higher temperatures, especially in small sections or cold-rolled material, causes the cementite to break up, and graphite is precipitated. Still higher temperatures cause all of the carbon to be taken into solution, and on cooling we have a very coarse grain with cementite envelopes and plates. Fig. 15 shows the corner of a small bar heated to 1,070° C. and slowly cooled. The carbon was 2 per cent. The outside is decarburized, and at the extreme corner we have a little ferrite, due to this cause. The interior, shown on the right, consists of globular cementite set in pearlite. Fig. 16 shows the same steel heated to 1,200° C. and slowly cooled. The ferrite is more pronounced than before, while the cementite now appears as envelopes around and in very coarse grains of pearlite. In the interior, much graphite has precipitated; and Fig. 17 shows the junction between the two areas: that on the right presenting cementite; that on the left, graphite. Fig. 21 shows a steel $\frac{5}{16}$ in. square, with 1.6 per cent. of C, heated to 1,200° C. and slowly cooled. Coarse envelopes of cementite surround the pearlite grains and the whole is very brittle.

Hardening is heating to above the line *SK* and suddenly cooling, as by quenching. The higher the temperature above *SK* the coarser the grain, and therefore the more brittle the material. Theoretically, if cooled quickly enough it should prevent any change in the solid solution austenite. Practically, this is impossible, and quenched material always shows more or less decomposition. Now this decomposition takes place in two ways. The outside of the grain of austenite may decompose into a dark-etching constituent, troostite; the nearer the outside of the bar, the less troostite. Fig. 22 shows a steel with 1.7 per cent. of C heated to white heat and quenched in water. The section is near the outside, and only a small amount of troostite appears. Fig. 23 shows a steel with 1.01 per cent. of C quenched from a yellow heat, section near the center of a bar $\frac{7}{8}$ in. square. The amount of troostite is now very great; only a small amount of austenite remains. The quenching-temperature being lower, the grain is proportionately smaller. The second method of decomposition occurs within the grain itself. Curious chevrons and laths appear, with some semblance of octahedral symmetry. This structure is called martensite. Fig. 24 shows it in a steel with 2 per cent. of C, $\frac{5}{16}$ in. square, quenched from 1,100° C. The thin white envelope to the very coarse grain is cementite. Between typical austenite with troostite, and martensite there is a whole range, depending on temperature of quenching, rate of quenching, and size of the section quenched. The grain-size depends on the quenching-temperature.

Tempering is reheating to from 200° to 300° C. to cause a further decomposition of the austenite, whereby brittleness is somewhat overcome, with some loss of hardness. Tempering does not alter the grain-size, but alters the structure of the grain. The martensitic structure becomes more and more pronounced and in time the ground-mass has changed from austenite into troostite and sorbite, and finally sorbite and pearlite are obtained, and then pearlite alone when the material has become completely softened. In order to follow the progress of tempering a small bar may be quenched at a bright yellow heat to get a coarse grain. One end is then heated to a dull red and the other end kept cold. On polishing this section the various changes can be traced from one end to the other. The maximum etching effect occurs at the part heated to about 400° C. According to Heyn, this osmonditic state should divide the stages troostite and sorbite, and coincides with the change in a number of properties.

VI. *Cementation and Case-Hardening.*

Under the microscope the process of case-hardening can be readily followed. It is best to examine the specimens without quenching, for cutting the specimen is much easier and the changes in structure are more easily recognized. On the outside is cementite plus pearlite, or pearlite alone; next ferrite makes its appearance in the pearlite, which gradually decreases in amount until finally the central zone is reached, where there is only the amount of pearlite in the original. Fig. 15 is the reverse of a case-hardened specimen, the position of the ferrite and cementite being reversed. If the material has been quenched it can be let down by heating to, say, 400° C. and cooling in air, when it may readily be cut with the saw. This will not affect the grain-size, and the core can thus be examined. A comparison between fracture and microstructure should always be made.

VII. *Alloy-Steels.*

The structure of alloy-steels forms a subject by itself. Alloy-steels have been classified into three groups, according to the predominant constituent: (1) pearlitic, (2) martensitic, and (3) austenitic. This, of course, refers to air-cooled specimens.

As an example of the first class is 3.5-per cent. nickel-steel, which closely resembles plain carbon-steel in structure. The grain-size is usually finer, and when quenched the martensite is very much stronger. Other alloy-steels, such as chrome-, nickel-chrome, chrome-vanadium, nickel-chrome-vanadium, silico-maganese, etc., all show pearlite. When quenched and tempered, however, there are profound differences both in properties and structure, martensite being predominant.

Manganese-steel, slowly cooled, shows a typical martensitic structure (Fig. 24), with more or less carbide around and between the grains. Quenching from 1,050° C. produces plain austenite, in polygonal grains. (See Fig. 13.)

Air-hardening and high-speed steels are martensitic, but the structure is by no means typical martensite. Small white grains of carbide are generally found in a complex groundmass, which at times resembles austenite, at others, martensite. The difference between properly hardened and improperly hardened steel can be recognized readily by those who have studied these steels with some care.

In conclusion, the correct application of metallography in practice ought to be emphasized. It does not compete with chemical analysis or physical testing; it supplements them. It tells about grain-size, segregation, inclusion of impurities, etc., and so is an excellent method for contrasting material and also for tracing the cause of failures. Some of its problems are of easy solution, as, for example, the distinction between properly and improperly refined castings, between wrought-iron and low-carbon steel, good and bad rail-steel, etc. Others, such as the study of hardened alloy-steels, especially high-speed steels, are difficult in the extreme, and require so much patience and skill that many of the workers along these lines have given up the pursuit after realizing its difficulties. As in all scientific work, however, application is rewarded sooner or later, and such subjects as the failure of steel wire in rope, at present unsolved, may in time be unraveled by metallographic methods.

SUBJECT TO REVISION.

[TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS.]

Notes on the Metallography of Alloys.

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(Cleveland Meeting, October, 1912.)

[SECRETARY'S NOTE.—To avoid repetitions of foot-notes, references to authorities are made in the paper by means of figures, referring to a numbered list in the appendix.—J. S.]

I. INTRODUCTION.

IN the olden days the making of alloys was an art, and the secrets of the craft were jealously guarded. To-day it has become a science, though the old ideas in regard to the secrets and formulæ are dying hard.

Modern progress may be attributed to scientific research and testing of materials, and the substitution of chemical analytical control for "cook-book" methods. We know to-day a great deal about the constitution of alloys, the reasons of the change in structure and physical properties by heat and mechanical treatment; and this new knowledge is due for the most part to the physical chemist and metallographist, who have shown us that alloys are simply solutions, and as such obey definite laws. The application of the theories of solutions and of the phase rule tells us what ought to happen when our alloys are in a state of equilibrium. The working-out of the temperature-composition diagram (thermal diagram) for numerous alloys has thrown a light on the structure and constitution which has dispelled most of the darkness in which we labored before the advent of metallography.

The work began with the determination of the freezing-point curves, and here the research of Le Chatelier,[1] Gautier,[2] Roberts-Austen,[3] and the other pioneers broke the ground. Next the physical chemists, Tammann,[4] Rooseboom,[5] Bancroft,[6] and their colleagues, showed us the complete thermal diagram and its meaning. Then the researches of Heycock and Neville[7] on the Cu-Sn series and of Shepherd[8] on the

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Cu-Zn demonstrated the application to the most complex of alloys.

A vast amount of work has been published on the constitution of alloys, as will be seen from a glance at the pages of the *Zeitschrift für anorganische Chemie*, *Zeitschrift für physikalishe Chemie*, *Metallurgie*, and *Revue de Métallurgie.* The diagrams have been collected notably by Portevin [9] in *Revue de Métallurgie*, Bornemann [10] in *Metallurgie*, and lastly in Guertler's *Text Book of Metallography*, now nearing completion.

The ternary diagram in the simplest case was worked out by Charpy [11] for the Pb-Sn-Bi series. More complicated cases have been given by Geer [12] and by Sahmen and Vegesack.[13]

The importance of bearing-metals in engineering makes no excuse necessary for a discussion of their constitution, and the following notes on the three groups of white metals (1) Pb-Sb-Sn, (2) Sn-Sb-Cu, (3) Zn base, and a comparison with the structure of the bronzes, may throw some light on the subject.

II. THE LEAD-TIN-ANTIMONY SERIES.

The ternary alloys of Pb-Sn-Sb find considerable application in the arts as bearing-metal, type-metal, white metal for small castings, etc.

In regard to the composition of alloys used in practice, the list given in Table I. embraces the most important. A few alloys containing a small amount of copper are also given for comparison. The list shows that Sb never exceeds 25 per cent. Pb varies from 93 to 5 per cent. and Sn from 1 to 75 per cent. There is no uniform variation, however.

The physical properties of this series of alloys are not very well known, at least they have not been published in detail, and while the anti-frictional qualities within certain limits of use do not show the wide differences one would expect from the variation in composition, there is probably a fairly marked difference in crushing-strength as we pass from the lead-rich to the tin-rich side, judging from the structure. Charpy, in a study of white alloys called anti-friction,[14] examined some alloys of the Pb-Sn-Sb group in regard to crushing-strength and microstructure.

Before taking up the ternary alloys a few notes on the binary alloys will be given.

TABLE I.—*Composition of Pb-Sn-Sb Alloys.*

No.	Name. Metal.	Pb.	Sn.	Sb.		Authority.
		P.C.	P.C	P.C.	P. C.	
1.	Electrotype-metal.,	93	3			Private notes.
2.	Bearing-metal	86	1			Private notes.
3.	Linotype-metal	85	3	12		Private notes.
4.	Bearing	83.3	8.3	8.3		Private notes.
5.	Stereotype-metal	82	6	12		Private notes.
6.	Bearing	82	2	16		Private notes.
7.	Stereotype	82	3.2	14.8		Roberts-Austen.
8.	Bearing	80	10	10		Private notes.
9.	Bearing—Compagnie de l'Est	80	12	8		Charpy. Guillet.
10.	Bearing	80	5	15		Guillemin. Guillet.
11.	Bearing—like "Glyco," etc	80.5	4.5	14.5	As, 0.5	Notes.
12.	Bearing—like "Magnolia"	78	6	16		Notes.
13.	Bearing—Magnolia and Tandem	77.7	5.9	16.8		Law. Thurston.
14.	Type metal	77.5	6.5	16		Notes.
15.	Bearing—Anti-friction	77	10	12.5	Cu, 0.5	
16.	Bearing—like "Coleco"	77	8	14	Cu, 1.0	
17.	Bearing	76	7	17		Notes.
18.	Metallic packing—Compagnie d'Orleans	76	14	10		Guillet.
19.	Bearing—American R. R	73.5	8	18.5		Notes.
20.	Piston Packing—Compagnie du Nord	73	12	15		Guillet.
21.	Bearing—French R. R	70	20	10		Charpy.
22.	Stereotype—Mackenzie metal	70	13	17		Thurston.
23.	Bearing—Paris-Lyon-Mediterranée R. R	70	10	20		Guillet.
24.	Type	70	10	18	Cu, 2	Roberts-Austen.
25.	Bearing—American R. R	68	21	11		Notes.
26.	Bearing—Graphite-metal	68	15	17		Dudley. Guillet.
27.	Stereotype	68	17	18		Thurston.
28.	Type	63.2	12	24	Cu, 0.8	Notes.
29.	Bearing	62.5	26.2	10	Cu, 1.3	Notes.
30.	Bearing	62	27	10		Notes.
31.	Type	60.5	14.5	24.2	Cu, 0.75	Notes.
32.	Type	60	35	5		Notes.
33.	Bearing	60	20	20		Brannt.
34.	Type—common	60	10	30		Law.
35.	Solder	60	39	1		Notes.
36.	Type	55.5	40	4.5		Notes.
37.	Type—best	50	25	25		Law.
38.	Bearing	48	40	10	Cu, 2	Notes.
39.	Bearing—American R. R., "No. 2"	46	36.5	16.5	Cu, 1	Notes.
40.	Hoyle's Metal	42	46	12		Hiorns. Guillet.
41.	Bearing. Chemin de fer de l'Est Francais	42	42	16		Guillet.
42.	Bearing	40	45	15		Ledebur. Guillet.
43.	Bearing—German	40	42	26	Cu, 2	
44.	Bearing—Italian R. R	37	38	25		Thurston. Guillet.
45.	Stereotype	35	60	5		Berthier. Thurston.
46.	White-metal	33	53	10.6	Cu, 2.4	Zn 1. Roberts-Austen.
47.	Bearing	10	75	15		Notes.
48.	Bearing	10	75	12	Cu, 3	Notes.
49.	For small castings	5	75	20		Notes.

1. *Lead-Antimony.*

Sb freezes at 631° C., Pb at 327° C. The addition of Pb to Sb, or of Sb to Pb, lowers the freezing-point. The freezing-point diagram showing the relation of percentage-composition to temperature of freezing was worked out by Roland-Gosselin,[15] who found that the eutectic or alloy with the lowest freezing-point contained Sb, 13 per cent., and froze at 228° C.

Stead,[16] in a paper on the microchemical examination of Pb-Sb alloys, showed that the eutectic contained Sb, 12.8 per cent. and froze at 247° C. Gontermann[17] gave 13 per cent. and 245° C.

The freezing-point or thermal diagram is shown in Fig. 1. Temperatures are represented in the vertical scale and percentage-composition in the horizontal. *A*, or 631° C., shows the freezing-point of Sb, which is lowered along the curve *Ar* as the proportion of lead in the alloy increases. Similarly, *B*, or 327 C., is the freezing-point of Pb, which is depressed along the curve *Br* as the antimony in the alloy increases. These two curves, indicating the freezing out of Sb and Pb respectively, intersect at the point *r*, or Sb, 13 per cent., at 247° C., the eutectic alloy, which is composed of a mechanical mixture of Pb and Sb due to their simultaneous freezing.

Thus alloys containing Sb from 100 to 13 per cent. are composed of crystals of Sb in an increasing amount of the eutectic or ground-mass, a type of structure which is shown in Figs. 6 and 8. At Sb, 13 per cent., the alloy is all eutectic and may be represented by Fig. 7. Alloys of Sb from 0 to 13 per cent. are composed of crystals of Pb in dendritic form set in an increasing amount of the eutectic. Fig. 9 represents this type.

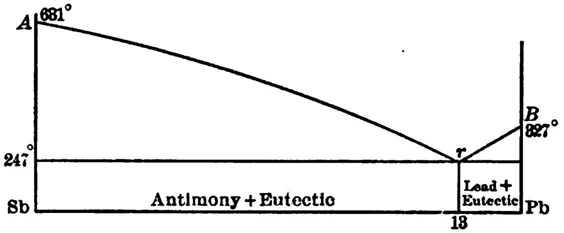

FIG. 1.—THE LEAD-ANTIMONY DIAGRAM.

In short, above the curve *ArB* the alloys are completely liquid, below the horizontal line at 247° C. everything is solid. In the left triangle, *Ar*—247° C., are crystals of Sb in a liquid alloy; in the right triangle crystals of Pb in a liquid.

The Pb-Sb alloys are used as bearing-metal, type-metal, hard lead, etc.

2. *Lead-Tin.*

The curve for Pb-Sn is similar to that of Pb-Sb. As determined by Kupffer[18] the eutectic contains Pb, 37 per cent., and freezes at 182° C. Charpy[14] gives Pb, 38 per cent., and Sn, 62 per cent. Roberts-Austen[19] gave the point at Sn, 68 per cent., and 180° C., and published curves showing tensile strength and extensibility. Tucker and Rosenhain[20] found the exact point to be Sn, 63 per cent., freezing at 182.5° C.

Thus alloys with Sn less than 63 per cent. are composed of crystals of Pb set in the eutectic: those with Sn more than 63 per cent. are composed of crystals of Sn set in the eutectic, Fig. 11. Solid Pb holds some Sn in solution, the amount varying with the rate of freezing. Rosenhain and Tucker were able to get 16 per cent. in solution by long annealing. As ordinarily made, however, the amount is very much less. Changes in the solid also occur.

The Pb-Sn alloys are used as solders, pewter, in the manufacture of toys, etc.

3. *Tin-Antimony.*

From a microscopic study of Sn-Sb alloys Charpy[21] found a definite compound containing Sb, about 50 per cent., which "formed isomorphous mixtures with Sb." He found (incorrectly) a eutectic at about 10 per cent. of antimony.

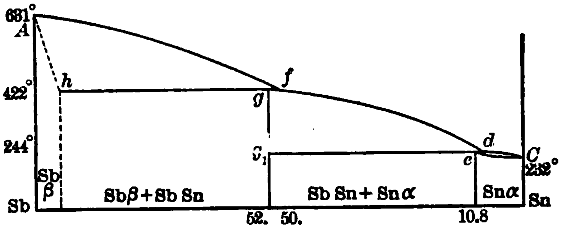

FIG. 2.—THE TIN-ANTIMONY DIAGRAM.

Stead[22] showed that with Sb up to 7.5 per cent. the alloys consist of a solid solution; above this point cubes of a compound Sb-Sn separate out when the alloy freezes. At the Sb end of the series crystals of Sb are the first to form.

The constitution of these alloys was not understood till Reinders[23] published his thermal diagram. Later Gallagher[24] modified the curve and showed a thermal transformation in the cubes (Sb-Sn of Stead) at about 315° C.

Williams[25] gave a simpler curve and omitted the transformation at 315° C. The main points of these three curves are the same, however, and Fig. 2 is taken from them, somewhat simplified, to explain the main changes.

The addition of Sn to Sb lowers the freezing-point progressively. The curve shows two inflections, at *f* and *d*, with two horizontals, *fgh* and deg_1. There is no true eutectic, as pure Sn freezes lower than any of the alloys of the series. Such a curve shows that above *AfdC* all is liquid: below $Ahgg_1eC$ all is solid. Between these two curves we have a mixture of solid and liquid. It also shows that at 422° C. the alloys with Sb from 90 to 50 per cent. show a reaction whereby the solid corresponding to *h* reacts with the liquid corresponding to *f* to form solid *g*. Now *h* is Sb containing Sn, 10 per cent., in solid solution, Sbβ for short; *g* corresponds to SbSn. The liquid *f* contains about 50 per cent. Similarly, at 244° C. we have the reaction of g_1 or Sb-Sn, with the liquid *d* (8 Sb) to form the solid *e* containing Sb, 10 per cent. (a solid solution of tin with Sb, 10 per cent., or *Sn α* for short). Then with Sb from 100 to 90 per cent. we have Sbβ. From 90 to 52 per cent. we have Sbβ surrounded by a ground-mass of Sb-Sn. From 52 to 10 per cent. we have Sb-Sn in a ground-mass of Sn *α*, Fig. 10, while from 10 to 0 per cent. we have Sn *α*, which is Sn containing Sb in solid solution.[26]

For the sake of simplicity the change in the Sb-Sn at 315° C. has been omitted, also the fact that Sb-Sn probably is capable of holding Sn (some 2 per cent.) in solid solution. In addition, while Sb dissolves Sn up to 10 per cent. at 422° C., the amount at ordinary temperature is much less.

The Sn-Sb alloys have been used as the basis of Britannia metal, for anti-friction alloys, for processes of engraving, etc.

4. *Ternary Alloys of Lead, Antimony, and Tin.*

The ternary diagram has been worked out by R. Loebe[27] and by Campbell and Elder.[28, 29]

The simplest method of representing a series of ternary alloys is by means of an equilateral triangle. In Fig. 3 the corners of the triangle represent the pure metals *A*, *B*, and *C*. The binary alloys of *A* and *B* are situated in the line *AB*, which is a projection of Fig. 1. Similarly, the binary alloys of *B* and *C* are represented by the line *BC*, and the alloys of *A* and *C* by the line *AC*, which is a projection of Fig. 2. An alloy of all three metals is represented by a point within the triangle, for the sum of the perpendicular distance from any point within the triangle to the three sides is constant and is taken as equal to 100. Take the alloy represented by *O* in Fig. 3: the percentage of *A* is given by the perpendicular from *O* to *BC*, the percentage of *C* by the perpendicular from *O* to *AB*, while the percentage of *B* is given by the perpendicular from *O* to *AC*. Hence, if we wish to indicate temperature, the scale must be perpendicular to the plane of the triangle; in other words, we must construct a solid figure.

The simplest case is where each pair of metals has a curve similar to Fig. 1. That is to say, the pairs are completely soluble in the liquid state and insoluble in the solid. Charpy[11] has worked out the Pb-Sn-Bi diagram and Stoffel those for Pb-Sn-Cd and Bi-Sn-Cd.

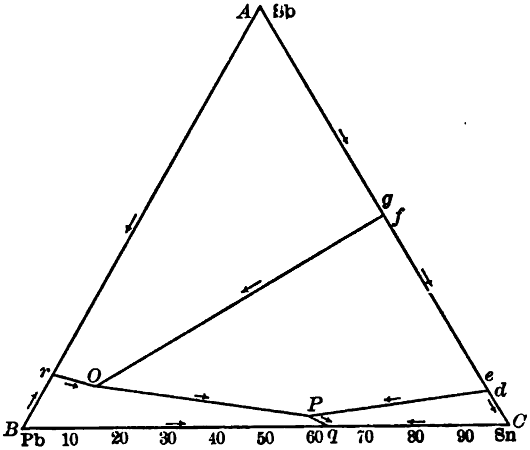

FIG. 3.—THE LEAD-TIN-ANTIMONY DIAGRAM.

More complicated cases have been worked out by Geer[12] and by Sahmen and Vegesack,[13] giving the theory. The latter show the case where the pairs *AB* and *BC* form eutectics and have curves represented by Fig. 1, while the pair *AC* shows no eutectic, but a reaction between the solid *h* and the liquid *f* to form the solid *g* (Fig. 2). Then at *O* (in Fig. 3) there is a similar reaction whereby solid *h'* and liquid *O* react, with the formation of solid *g'* and *B*.

In the case of Pb, Sn, and Sb, there are two pairs represented by Fig. 1, and the third by Fig. 2, where there are two reactions. Fig. 3 gives such a ternary diagram, and the following is a brief description of the changes that take place when an alloy solidifies: *A* represents Sb, *B* is Pb, and *C* is Sn. Then *r* is the eutectic of Pb and Sb, with Sb, 13 per cent., freezing at 247° C. Again, *q* is the lead-tin eutectic at 63 per cent. of tin, freezing at 182° C. On the side *AC* the point *g* represents Sb-Sn, *f* the reaction-point at 422° (*f* in Fig. 2), *d* the reaction-point at 244° (*d* in Fig. 2), and *e* is Sb, 10 per cent., or saturated Sn *a*.

In the area *ArOf* the first solid to form is Sb (with a little Sn in solid solution). In the area *BrOPq*, the Pb crystallizes out first. In *fOPd*, Sb-Sn forms first; while in *CdPq*, the Sn crystallizes out first, with more or less Sb in solid solution (Sn *a*).

Assuming, for the sake of simplicity, that pure Sb crystallizes out, in the triangle *AOr* an alloy will deposit crystals of Sb on freezing until the composition of the liquid reaches a point on the line *rO*. Then we shall have Pb and Sb freezing out side by side, and the liquid composition sliding down the line *rO* till it reaches the reaction-point, *O*. At this point occurs the reaction, Sb crystals + liquid *O* forming Sb-Sn + Pb. If the liquid is used up before the Sb, the alloy goes solid at *O*. If not, when the Sb has disappeared, the liquid follows the line *OP*, and the Sb-Sn and Pb separate out side by side till the alloy finally is all solid.

In the area *BOr*, the first to freeze are crystals of Pb, and the liquid reaches the line *rO*, the further changes being as above.

In the area *AOf*, the Sb crystals freeze out, and the liquid reaches the line *fO*, and follows down it towards *O*. This change in composition of the liquid is brought about through a reaction between the solid Sb and the liquid to form Sb-Sn. On reaching *O*, as before, we have the reaction above mentioned, but if the Sb is all gone before the liquid reaches *O*, further solidification occurs as in the next case.

In the area *fPO*, the Sb-Sn crystallizes, and the liquid reaches the line *OP* and follows it, due to the simultaneous freezing of Sb-Sn and Pb. On reaching the point *P*, we have another reaction, similar to the one at *O*. Here Sb-Sn + liquid *P* react to form solid Sn *a* + Pb. As before, if all of the Sb-Sn is used up before the liquid *P*, further freezing follows the line *Pq*, and the alloy finally becomes solid.

In the area *BOP*, the Pb freezes out first, and the liquid, on reaching line *OP*, follows it, as above.

In the area *BPq*, the Pb crystallizes out first, and on the liquid reaching the line *Pq*, the Pb and Sn (Sn *a*) freeze out side by side till *q* is reached, the binary eutectic of Pb and Sn, also the eutectic of the system.

In the area *fPd* crystals of Sb-Sn are the first to form. When the liquid reaches the line *dP* this stops and we get the reaction Sb-Sn + liquid = Sn *a*, which continues as the liquid follows the curve *dP* to *P*. If the liquid uses up all of the Sb-Sn before *P* is reached, further freezing takes place as in the next case. If not, then at *P* occurs the reaction Sb-Sn + liquid *P* = Sn *a* + Pb and the alloy finally becomes solid.

In the area *CdPq*, the tin solid solution (Sn *a*) first freezes out and then the liquid reaches the line *Pq* as before.

In short, the line *rO*, the boundary between the Sb area and the Pb area, represents the simultaneous freezing of Pb and Sb. The line *OP* is the boundary between the Pb and the Sb-Sn areas and represents the simultaneous freezing of Pb and Sb-Sn; while *Pq* is the boundary between Pb and Sn *a* and represents the simultaneous freezing of these two. All of these give structures resembling that of a true eutectic.

On the other hand, the boundary-line *fO* represents the reaction whereby Sb-Sn forms at the expense of the Sb and *dP* represents the reaction whereby Sn *a* forms at the expense of Sb-Sn. These two points are:

(1) *O* at, Pb, 80; Sn, 10; Sb, 10 per cent., at 245° C.
(2) *P* at, Pb, 40; Sn, 57.5; Sb, 2.5 per cent., at 189° C.

5. *Microstructure.*

In the preparation of sections for the microscope the pieces must be sawed through vertically because there is often a marked difference between the top and the bottom of the specimen. This is due to the fact that both the Sb and the SbSn crystals are much lighter than the liquid out of which they freeze and they therefore tend to float to the surface of the melt. After cutting with a saw the surface of the section must be carefully filed to take off all of the surface altered in the process of cutting. If this is not done the structure will be broken up and obscure. After polishing in the usual way the final polish on a rouge-board must be continued longer than usual till a good surface is obtained. As an etching reagent 2 per cent. of nitric acid in alcohol is very good.

With small melts cooled in the air the structure shows that equilibrium is not always established. Alloys in that part of the area *ArOf* the Sb of which ought all to have disappeared, due to the reaction on *fO* or at *O*, often show undissolved cores of Sb. Similarly with Sb-Sn in the area *fOPd*. With very slow cooling, of course, conditions would have brought about more complete equilibrium.

The microstructure of the series is typically shown in the following photographs:

Fig. 6. Pb, 65; Sb, 25; Sn, 10 (× 45). The white crystals are Sb set in a ground-mass which freezes at *O*, a mixture of Sb-Sn and Pb (Fig. 7).

Fig. 7. Pb, 78; Sb, 11; Sn, 11 (× 90). Fig. 8 (the same × 45) occurs very near the point *O*. Fig. 8 is the top of the alloy and shows the layer of Sb-Sn cubes which have floated to the surface. The structure of the ground-mass is well shown in Fig. 7 and is a mixture of Pb and Sb-Sn.

Fig. 9. Pb, 85; Sb, 10; Sn, 5 (× 56). The black dendrites are Pb surrounded by the same ground-mass as before.

Fig. 10. Pb, 20; Sb, 10; Sn, 70 (×45). White cubes of Sb-Sn set in a background of Sn *a*, which separates out along *dP*, and a dark ground-mass of Pb-Sn *a* which froze at *P* or on *Pq*.

Fig. 11. Pb, 37.5; Sb, 2.5; Sn, 60 (× 55). The dendrites are Sn (Sn *a*) surrounded by the Pb-Sn *a* ground-mass as above.

III. ALLOYS OF TIN, ANTIMONY, AND COPPER.

These alloys find extensive use as bearing-metal, Britannia metal, etc. Table II. gives a number of typical analyses from various sources and shows that usually Sn exceeds 70, Sb is less than 20, and Cu less than 10 per cent. Charpy[14] examined about 20 alloys for microstructure and compressive strength. Before discussing the ternary alloys a word or two must be said of the Cu-Sn series.

TABLE II.—*Composition of Sn-Sb-Cu Alloys.*

No.	Name. Metal.	Sn.	Sb.	Cu.		Authority,
		P.C.	P.C.	P.C.	P. C.	
1.	English Britannia	94	5	1	....	Law.
2.	Bearing	91	4.5	4.5	....	Private notes.
3.	English Britannia—sheet	90.6	7.8	1.5	....	Brannt.
4.	English Britannia—cast	90.6	9.2	0.2	....	Law.
5.	Bearing	90	6	4	....	Private notes.
6.	Bearing—Russian R. R.	90	8	2	....	Thurston.
7.	English Britannia	90	6	2	Bi, 2	Law.
8.	English Britannia	90	7	3	....	Law.
9.	Bearing	89.3	8.9	1.8	....	Thurston.
10.	Pewter	89.3	7	1.8	Pb, 1.8	Brannt.
11.	Bearing	88.9	7.4	3.7	....	Karmarsch. Bolley.
12.	Bearing	88.8	7.5	3.7	....	Law. Thurston.
13.	Queen's metal	88.5	7.1	3.5	Zn, 0.9	Law.
14.	Queen's metal	88.5	7	3.5	Bi, 1.0	Law.
15.	Bearing	87	7	6	....	Hiorns.
16.	English Britannia	85.5	9.7	1.8	Zn, 3	Law.
17.	Bearing—heavy	85	7.5	7.5	....	Private notes.
18.	Jacoby metal	85	10	5	....	Ledebur. Brannt.
19.	German Britannia	84	9	2	Zn, 5	Law. Brannt.
20.	French car-bearings	83.3	11.1	5.5	....	Charpy.
21.	Bearing	83.3	8.3	8.3	....	Private notes.
22.	Bearing—German R. R.	83	11	6	....	Ledebur.
23.	Bearing—valve-rods, etc.	82	10	8	....	Law.
24.	Bearing—French R. R.	82	12	6	....	Ledebur.
25.	Britannia (Baumgartel)	81.9	16.3	1.8	....	Bolley. Brannt.
26.	Bearing—Swiss R. R.	80	10	10	....	Thurston.
27.	Ashberry metal	80	14	2	Zn, 1	Law.
28.	Ashberry metal	79	15	3	Zn, 2	Law.
29.	Britannia. Ashberry	77.8	19.4	2.8	....	Thurston. Bolley.
30.	Britannia. Ashberry	77.9	19.4	......	Zn, 2.8	Roberts-Austen.
31.	Bearing—English	76.7	18.5	7.8	....	Brannt. Bolley.
32.	Bearing—German	76	17	7	....	Brannt.
33.	Bearing	73	18	9	....	Thurston.
34.	Bearing	72	26	2	....	Thurston. Bolley.
35.	German Britannia	72	24	4	....	Brannt. Law.
36.	Bearing (Karmarsch)	71.4	7.2	21.4	....	Thurston. Bolley.
37.	Bearing—Valve packing	71	24	5	....	Thurston.
38.	Bearing (Karmarsch)	70.7	19.7	9.5	....	Thurston. Bolley.
39.	Minofor (Britannia)	68.5	18.2	3.3	Zn, 10	Brannt. Law.
40.	Bearing—G. W. R. (England)	67	11	22	....	Thurston.
41.	Bearing—French R. R.	67	22	11	....	Law. Charpy.
42.	Dewrance metal (locomotive)	33.3	44.5	22.2	....	Charpy. Hiorns.

1. *Copper and Tin.*

In Fig. 4 is given a simplified diagram. Some of the changes have been omitted (*e.g*, that which occurs in the alloys between *c* and *C* at 740° C.) because the various researches, notably by Heycock and Neville, Shepherd and Blough,[30] Giolotti and Tavanti,[31] do not agree, and these points are still unsettled. The freezing-point curve, or liquidus, is *ABCDEFG*. The solidus, or end of freezing, is given by *AabcdxnefFg*. In alloys with Sn from 0 to 38.2 per cent. (Cu_3Sn) the curves below the solidus represent changes in the solid alloy.

Considering the Cu-Sn thermal diagram in small parts it at once becomes simple, being merely compounded of a number of simple curves.

The Two Solid Solutions, Alpha and Beta.

The part of the diagram embraced by *ABC*—*Aabc* and just below it, shows that the conditions are similar to that represented by Fig. 2.

In freezing, a solid solution rich in Cu (alpha or the solid *I*) separates out when the temperature reaches the curve *AB*. This enriches the liquid in Sn, and at 795° C. there are dendrites of alpha, saturated with Sn of composition *a*, and a liquid of composition *B*. Then occurs a reaction: solid solution alpha (*a*) + liquid B = solid solution beta (*b*). This reaction continues till all of the alpha or all of the liquid is used up and the alloy

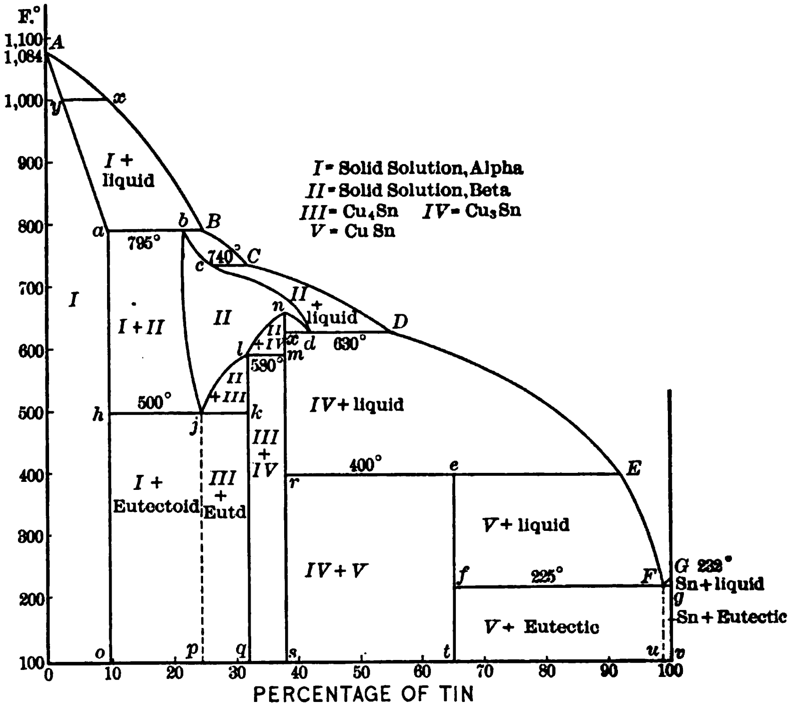

FIG. 4.—THE COPPER-TIN DIAGRAM.

becomes solid. In the area marked by the solid *I* is the solid solution rich in copper, or alpha. In the triangular area marked by the solid *II* is nothing but the beta solid solution, while between these two areas is *I* + *II*, or the mixture of alpha and beta (see Figs. 18 and 24).

But this beta solid solution, which contains Sn, 22 per cent., at 795° C., becomes enriched in Sn as the temperature falls, and this it does by following down the curve *bj* and precipitating some alpha. On arriving at *j*, the eutectoid point at 500° C., it suddenly breaks up into a mechanical mixture of alpha and another solid which we will call Cu_4Sn (although it may be merely a solid solution and not a compound). This mechanical mixture is the eutectoid and corresponds to pearlite in steel.

Beta Alone.

In alloys between *b* and *p* (Sn, from 22 to 25 per cent.) the beta solution in the area *II* in cooling reaches the curve *hj* and alpha separates out, enriching beta in tin until at 500° C. it corresponds to *j*, or Sn, 25 per cent., when it splits up into the eutectoid as before.

In alloys between *p* and *q*, or Sn, 25 and 31.8 per cent. respectively, the beta solution in the area *II*, on cooling down, reaches the curve *lj*, and *III*, or Cu_4Sn, starts to precipitate. Thus in area *ljk* are grains of beta surrounded by a network of Cu_4Sn, just as in high-carbon steel the solid solution (austenite) can occur surrounded by iron carbide (cementite). At 500° C., or *hjk* line, the remaining beta of composition *j*, or Sn, 25 per cent., splits up into the eutectoid as before.

Therefore into the normal state we have:

Sn from 0 to 10 per cent. (alpha solid solution). Sheet-bronze to gun-metal.

Sn from 10 to 25 per cent. (alpha grains set in eutectoid, Figs. 24 and 19). Gun-metal to bell-metal.

Sn from 25 to 31.8 per cent. (Cu_4Sn and eutectoid, see Fig. 12).

Sn 31.8 per cent. (pure Cu_4Sn). Speculum-metal.

Limit of the Alpha Solid Solution.

The amount of Sn that Cu holds in solid solution depends on the rate of freezing and cooling. In an ordinary casting, when Sn exceeds 6 per cent., traces of the eutectoid can be seen. At Sn, 10 per cent. (a gun-metal), the structure closely resembles Fig. 24, the euctectoid being quite noticeable. In these alloys, however, the alpha grains are not saturated. Long annealing gives us homogeneous alpha grains, up to Sn, 12 or 13 per cent. Thus sheet gun-metal is all alpha.

The eutectoid is hard, sonorous, but somewhat brittle. Thus for small bells we can use almost the eutectoid alloy, while for larger ones we must have more of the alpha grains, which decrease the brittleness and lower the tone. Fig. 19 might represent the structure of ordinary bell-metal.

The compound Cu_4Sn is very brittle, but takes a high polish and is used for mirrors in optical work.

The Compounds Cu_4Sn to Cu_3Sn, or Sn, 31.8 *to* 38.2 *Per Cent.*

Taking the area *II* above the curve *ln*, the beta solid solution, on cooling down, reaches *nl*, and the compound Cu_3Sn, or *IV*, precipitates out. Thus in the area *mln* we have a mixture of beta and Cu_3Sn. Now at 580° C., or *lm*, the remaining beta changes into the compound Cu_4Sn, so that below 580° C. these alloys consist of Cu_3Sn and Cu_4Sn, *i. e.*, *III* plus *IV*. They are very brittle and not commercially useful.

Alloys Rich in Tin.

The alloys with Sn in excess of 38.2 per cent. are all brittle and of no commercial importance until those containing about 85 per cent. are reached. This part of the curve shows two horizontals, *Dx* and *Eer*, besides the eutectic line *fFg*. At 630° C. the line *Ddx* gives the temperature of the reaction solid solution *II* + solid *IV*, or Cu_3Sn + liquid *D*. Similarly, the line *Eer* at 400° C. represents the reaction: *IV* (Cu_3Sn) + liquid $E = V$ (CuSn). Theoretically this reaction ought to be complete, and in alloys with Sn from 38.2 to 65 per cent. the liquid is all used up before the Cu_3Sn, and therefore the kernels of Cu_3Sn are surrounded by a ground-mass of hard white Cu-Sn, or *IV* + *V*. In practice, however, the Cu-Sn forms envelopes round the Cu_3Sn and the reaction is not complete, so in the area *IV* + *V* occurs Cu_3Sn + Cu-Sn and some eutectic, while in alloys with Sn from 65 to 92 per cent., *e* to *E*, are more or less cores of Cu_3Sn inside the Cu-Sn set in the excess eutectic, Fig. 15. Theoretically all of the Cu_3Sn should be used up (and is, on very slow cooling) before the liquid and the resulting alloys should be Cu-Sn (*V*) + eutectic.

On the right hand side of the diagram is the *V* curve *EFG;* this is similar to that of Fig. 1, for *EF* denotes the freezing-out of Cu-Sn in laths and needles, *FG* the freezing of crystals of Sn in dendrites, while the horizontal *gFf* is the freezing of the eutectic of Sn and Cu-Sn. See Figs. 14, 13, and 12. The Cu-Sn needles, skeleton stars, etc., are quite characteristic, and together with the CuSn-Sn eutectic enter into the constitution of tin-rich babbitt metals.

The horizontal *Een* illustrates the case of the compound, which dissociates before reaching the melting-point; thus, on heating, Cu-Sn, or *V*, splits up at 400° C. into Cu_3Sn (*IV*) + liquid *E*.

The eutectic point *F* is Cu, 1 per cent., and 225° C. (Fig. 13). In the alloys with less than Cu, 1 per cent., there are grains and dendrites of Sn, surrounded by the eutectic (Fig. 12). Thus, in the useful alloys are:

Cu from 15 to 8 per cent. Cores of Cu_3Sn, envelopes, and needles of Cu-Sn set in the eutectic. Fig. 15.

Cu from 8 to 1 per cent. Cu-Sn needles, stars, etc., in the eutectic. Fig. 14.

Cu from 1 to 0 per cent. Dendrites and grains of tin, surrounded by the eutectic of Sn and Cu-Sn. Fig. 12.

2. *Ternary Alloys of Tin, Antimony, and Copper.*

In Fig. 5 is given an approximate diagram of Sb up to 30 per cent., and Cu, 30 per cent., worked out from microstructure. The changes which take place are similar to those of the Pb-Sb-Sn series.

The line *cq* represents the reaction between Cu_3Sn and the liquid to form Cu-Sn. The line *do* is the reaction between Sb-Sn and the liquid to form Sn α. The line *po* is the simultaneous freezing of Sb-Sn and Cu-Sn, while *ob* is the simultaneous freezing of Sn α and Cu-Sn, giving a eutectic-like structure, as in Fig. 13.

The point *O* is approximately Cu, 1; Sb, 7.5; Sn, 91.5 per cent., and represents the reaction: Sb-Sn + liquid o = Sn α + Cu-Sn.

Any alloy on the left of *qc* will freeze out Cu_3Sn, and when the liquid reaches the line *cq*, the reaction to form Cu-Sn occurs. Now in ordinary cooling the Cu_3Sn is soon covered by Cu-Sn (Fig. 15), and the reaction stops, while Cu-Sn continues to freeze out normally from the liquid. The further structure of the alloy depends on whether the liquid reaches the line *po* or *ob*. Suppose the liquid reaches *po*, then there is a simultaneous separation of Cu-Sn and Sb-Sn, following *po*, till at *O* occurs the reaction to form Sn *a* + Cu-Sn. Again, the reaction is not completed, and the composition of the liquid follows *ob* till the whole is solid. This is shown in Fig. 17: Sn, 67; Sb, 11; Cu, 22 (× 45). The dark cores to the plate-like crystals are Cu_3Sn. The white envelopes and dots are Cu-Sn, the white cubes are Sb-Sn, and the ground-mass is a mechanical mixture of Cu-Sn and Sn *a* (see Fig. 13), which freezes along *ob*.

The case where the liquid reaches *ob* is simpler, for it freezes along *ob* as before and the alloy shows no cubes of Sb-Sn. Fig. 15, Sn, 70; Sb, 5; Cu, 25 (× 45), shows the dark cores of Cu_3Sn surrounded by the white envelopes of Cu-Sn, in a ground-mass which freezes along *ob*.

Coming next to those alloys the composition of which falls within *pobcq*, the first thing to freeze out is Cu-Sn, thus enriching the liquid in Sn and Sb, until it arrives at *po* or *ob*. The further changes are of two kinds, as above. Fig. 16, Sn, 83.5; Sb, 11; Cu, 5.5 (× 45), started to freeze by separating needles and skeleton crystals of Cu-Sn. The liquid finally reached the line *po* and then occurred a simultaneous freezing of cubes of Sb-Sn and needles of Cu-Sn until the point *O* was reached. Then followed reaction and solidification along *ob* as before. Fig. 14, Sn, 90; Sb, 4.5; Cu, 4.5 (× 50), began to freeze out Cu-Sn as before, but the liquid reached the line *ob* and followed along it till solid. Therefore no cubes appear.

Any alloy in the area *pode* first freezes out cubes of Sb-Sn. These are lighter than the liquid out of which they freeze, and float to the surface, as shown in Fig. 8. The liquid may reach *op* or *od*. If *op*, then Sb-Sn cubes and Cu-Sn needles crystallize out together, and freezing continues, as already described. If *od*, the cubes tend to react with the liquid to form Sn *a* till *o* is reached. This reaction is never completed in ordinary cooling, and further freezing follows *ob*, as before.

The reason why the cubes in Fig. 16 have not floated to the surface is because they are held down by a skeleton of Cu-Sn crystals, which froze before or with them. This is well shown in Fig. 17.

Alloys on the line *ob* are represented by Fig. 13: Sn, 94; Sb, 5; Cu, 1 (× 50). Here we see Sn *a* and Cu-Sn separating simultaneously, and we get a pseudo-eutectic structure.

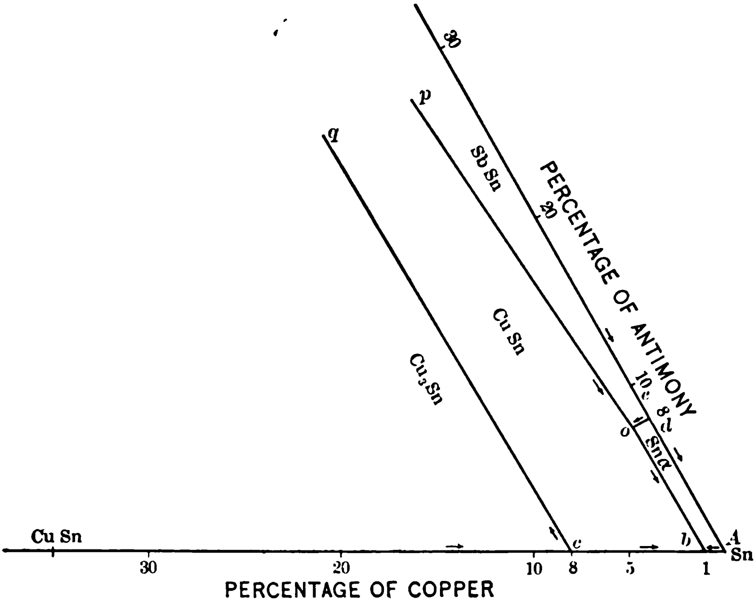

Fig. 5.—Part of the Tin-Antimony-Copper Diagram.

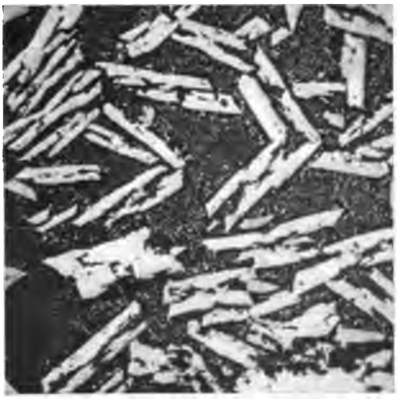

Fig. 6.—Pb, 65 ; Sb, 25 ; Sn, 10. × 45.

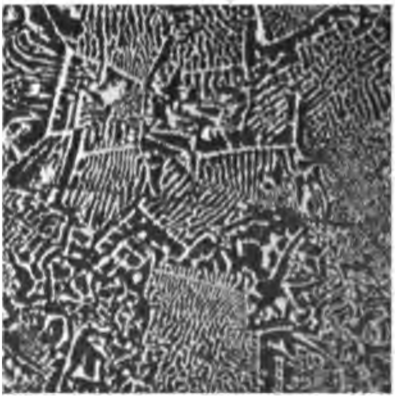

Fig. 7.—Pb, 78 ; Sb, 11 ; Sn, 11. × 90.

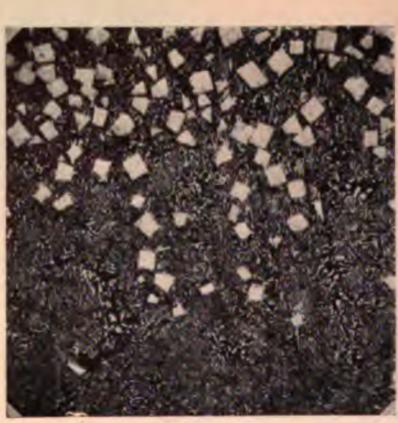

FIG. 8.—Pb, 78; Sb, 11; Sn, 11. × 45.

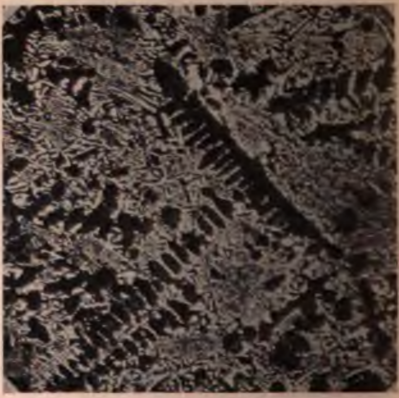

FIG. 9.—Pb, 85; Sb, 10; Sn, 5. × 56.

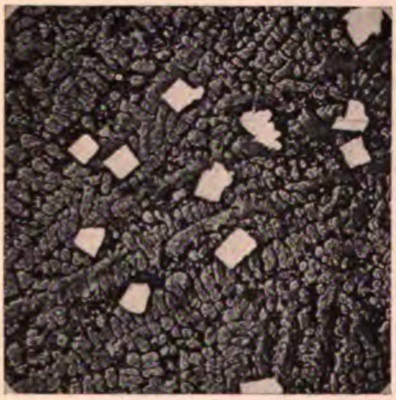

FIG. 10.—Pb, 20; Sb, 10; Sn, 70. × 45.

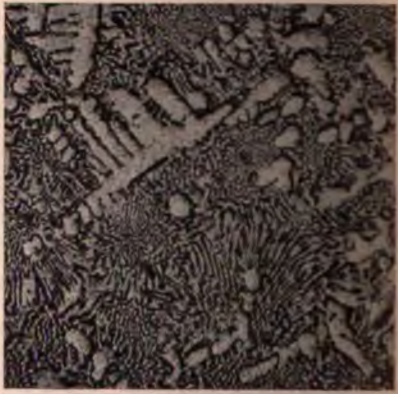

FIG. 11.—Pb, 37.5; Sb, 2.5; Sn, 60. × 55.

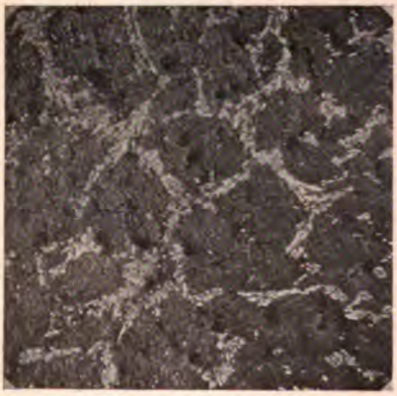

FIG. 12.—Sn, 94.5; Sb, 5; Cu, 0.5 × 45.

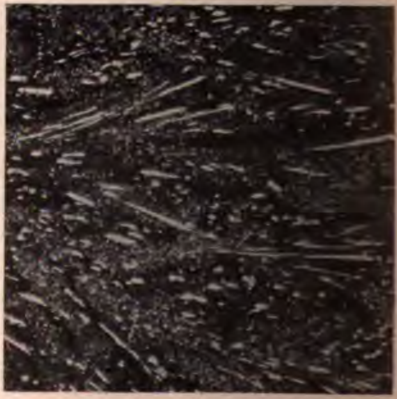

FIG. 13.—Sn, 94; Sb, 5; Cu, 1. × 50.

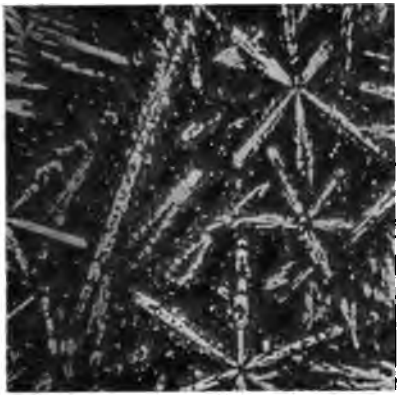

Fig. 14.—Sn, 90; Sb, 4.5; Cu, 4.5. × 50.

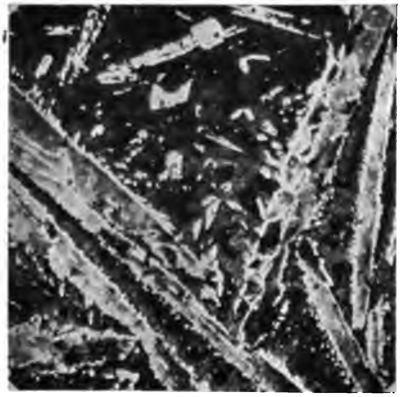

Fig. 15.—Sn, 70; Sb, 5; Cu, 25. × 45.

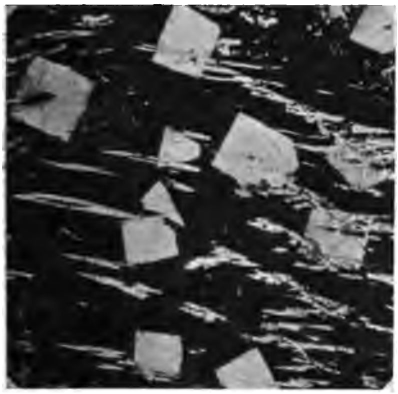

Fig. 16.—Sn, 83.5; Sb, 11; Cu, 5.5. × 45.

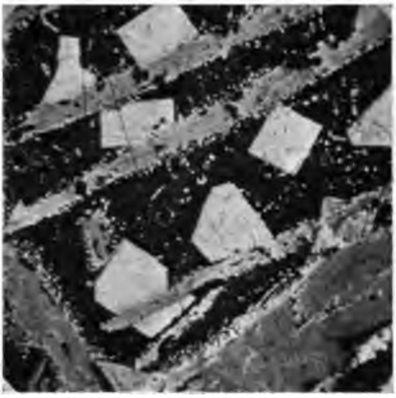

Fig. 17.—Sn, 67; Sb, 11; Cu, 22. × 45.

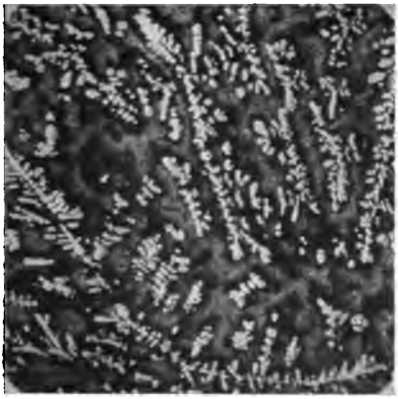

Fig. 18.—Zn, 96; Cu, 4. × 45.

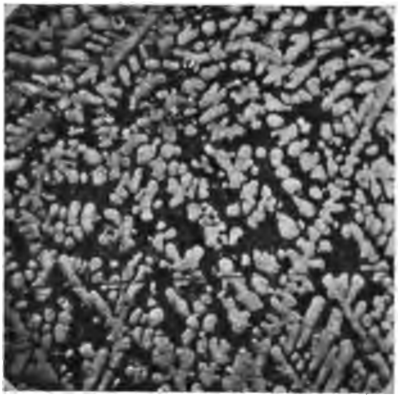

Fig. 19.—Zn, 85; Cu, 10; Al, 5. × 55.

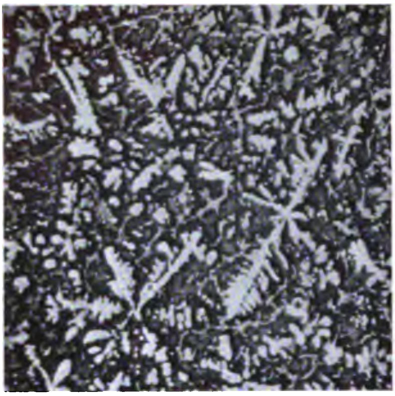

FIG. 20.—ZN, 85; CU, 5; SN, 10. × 45.

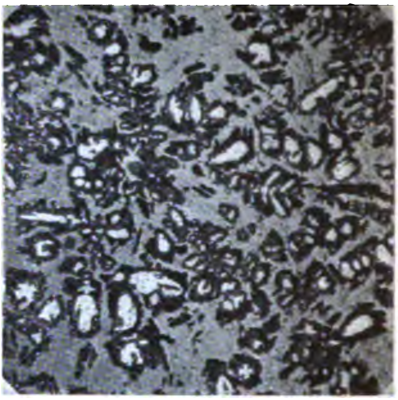

FIG. 21.—ZN, 35; SN, 60; CU, 5. × 45.

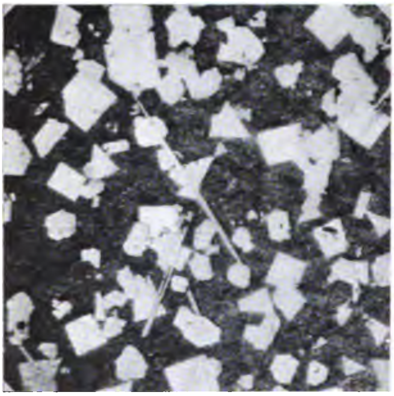

FIG. 22.—SN, 78; SB, 19; ZN, 3. × 45.

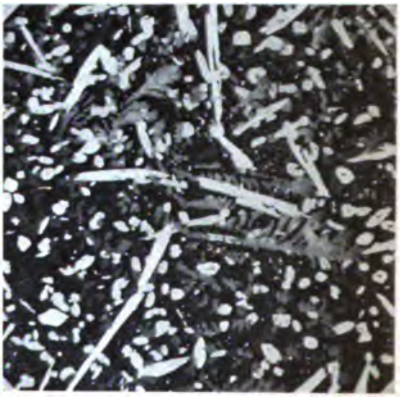

FIG. 23.—ZN, 85; SB, 10; CU, 5. × 45.

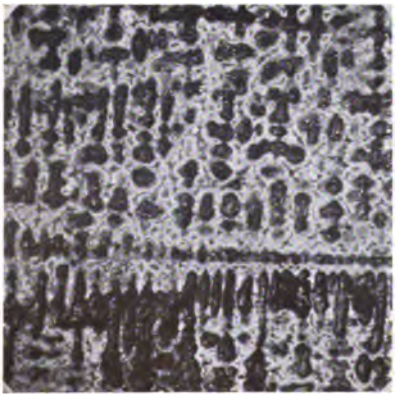

FIG. 24.—CU, 88; SN, 10; ZN, 2. × 45.

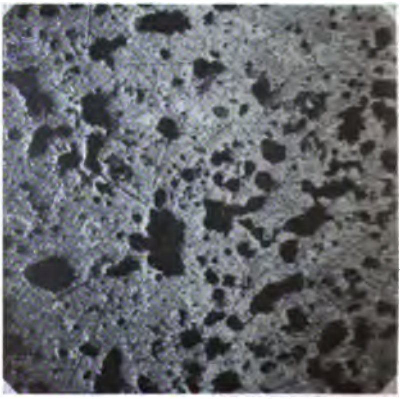

FIG. 25.—CU, 62; SN, 11; PB, 27. × 65.

Lastly, alloys within the area *bodA* consist of dendrites and grains of Sn α set in this pseudo-eutectic, which freezes along *ob*. Fig. 12 shows the alloy Sn, 94.5; Sb, 5; Cu, 0.5 (× 95). The dark etching grains are the Sn α surrounded by the lighter ground-mass, a mixture of Sn α and Cu-Sn.

From the above, it will be seen that in this series most of the alloys are not in a state of equilibrium, because the various reactions are incomplete. As the copper-content is increased the amount of Cu-Sn increases, till, finally, Cu_3Sn is present. This increase in Cu increases the hardness (and brittleness) and decreases the plasticity proportionately. Similarly, an increase in Sb increases the amount of the hard cubes of Sb-Sn, with similar results.

IV. ZINC-BASE ALLOYS.

Alloys rich in Zn have an extensive use as bearing-metals. Most of them contain Cu as well as Sn. Table III. gives the analyses of a number of alloys which have been used or advocated.

TABLE III.—*Composition of Zinc-Base Alloys.*

No.	Name. Metal.	Zn.	Cu.	Sn.	Sb.		Authority.
		P.C.	P.C.	P.C.	P.C.	P. C.	
1.	Hamilton metal	93.4	3.5	......	1.5	Pb, 3.1	Thurston. Bolley.
2.	Bearing—hard	90	7	1.5	1.5	....	Brannt.
3.	Ehrhardt's metal	89	4	4	......	Pb, 3	Thurston. Bolley
4.	Bearing	88	8	2	2	....	Brannt.
5.	Salge metal	85.5	4	9.9	......	Pb, 1.1	Dudley.
6.	Lumen	86	10	......	......	Al, 4	Metal Industry.
7.	Lumen	85	10	......	......	Al, 5	Notes.
8.	Bearing	85	5	......	10	....	Ledebur.
9.	Bearing	85	5	10	......	....	Notes.
10.	Biddery metal—Heine's	84.3	11.4	1.4	......	Pb, 2.9	Bolley. Brannt.
11.	Pierrot metal—Beugnot	83.3	8.3	7.6	3.5	Pb, 3	Ledebnr.
12.	Fenton's alloy	80	6	14	......	....	Law.
13.	Fenton's alloy	80	8.5	......	14.5	....	Thurston. Bolley.
14.	Bearing	77	5.5	17.5	......	....	Ledebur.
15.	English white metal	76.2	5.6	17.5	......	Pb, 0.7	Brannt.
16.	Vaucher's alloy	75	......	18	2.5	Pb, 4.5	Hiorns.
17.	For pump-cocks.	72	7	21	......	....	Bolley.
18.	For propeller-bushing	69	5	26	......	....	Roberts-Austen.
19.	Babbitt metal	69	5	19	3	Pb, 5	Ledebur.
20.	Bearing—English	67.7	7.4	14.9	......	....	Thurston. Bolley.
21.	Bearing	66.5	4.2	29.3	......	....	Thurston. Bolley.
22.	Dunnlevic & Jones metal	52	1.6	46	0.4	....	Charpy.
23.	Heavy axle	47	1	38	6	Pb, 4	Brannt.
24.	Heavy axle	40	3	15	......	Pb, 3.2	Brannt.
25.	Parsons white brass	30	5	65	......	....	Notes.

The binary alloys of Zn and Cu show at the Zn end of the series (Shepherd,[8] Tafel[32]) two solid solutions, ϵ, with Cu, 14 per cent., and η, with Cu, 2.5 per cent. There is no eutectic, but a reaction-point, as in the alloys rich in Sn in Fig. 2. Thus, Fig. 18 (× 45) is the alloy with Zn, 96; Cu, 4 per cent. The white dendrites are the ε with Cu, 14 per cent., and the dark etching ground-mass is η with Cu, 2.5 per cent., in solid solution. Now, most of the zinc-base bearing-metals show dendrites of ε set in a ground-mass, which varies with the third metal.

1. *Zinc and Tin.*

The Zn-Sn alloys can be shown by Fig. 1. The eutectic occurs at Zn, 8 per cent., and 204° C. As the Zn end of the Cu-Zn diagram can be represented by the right-hand side of Fig. 2, the zinc-rich corner of the ternary diagram can be explained by the right-hand corner of Fig. 3, if d represents Cu, 2.5 per cent., and q, Sn, 92 per cent., the C would be pure zinc. For example, Fig. 20 is Zn, 85; Cu, 5; Sn, 10 (× 45). The white dendrites are ε (Cu, 14; Zn, 86), and crystallized out until the liquid reached the line dP. Following this line there was the reaction between ε and the liquid to form η, which continued until at P the rest of the liquid went solid, for $\epsilon + \text{liquid} = \eta + \text{Sn}$. The latter can be seen as irregular light lines in the ground-mass.

Again, Fig. 21: Zn, 35; Sn, 60; Cu, 5 (× 45). As before, the white dendrites are ε, surrounded by dark envelopes, the η, while the ground-mass is the matrix rich in Sn, which froze at P, as before.

2. *Zinc and Aluminum.*

The metals Zn and Al were thought to form no compound (Shepherd[38]), but Rosenhain and Archbutt[39] show that the eutectic Al, 5 per cent., at 380°, is composed of a mechanical mixture of Zn and Al_2Zn_3.

Then in the alloy shown in Fig. 19, Zn, 85; Cu, 10; Al, 5 (× 55), the white dendrites are ε, as before, but the ground-mass, which freezes mainly at the point P (Fig. 2), is composed of η and Al_2Zn_3. Under high powers it is clearly such a mechanical mixture.

3. *Other Alloys of Zinc.*

Two other alloys are given as representing a different form of crystallization. Fig. 23, Zn, 85; Sb, 10; Cu, 5 (× 45), the bright needles are Sb_2Zn_3, set in a complex ground-mass. Fig. 22 is an alloy rich in Sn—Sn, 78; Sb, 19; Zn, 3 (× 45). There are three distinct kinds of crystals: the bright white cubes of

Sb-Sn, darker cube-like forms, perhaps Sb-Zn, with a few needles of Sb_2Zn_3, the whole set in a ground-mass rich in tin.

To make a general summary of the structure of these white anti-frictional alloys, they are composed of hard crystals or dendrites, such as Sb-Sn, Cu_3Sn, Cu-Sn, Sb_2Zn_3, ϵ, etc., set in a more or less plastic ground-mass.

V. THE BRONZES.

For the sake of comparison between the white metals and the bronzes, two micrographs are given, Figs. 24 and 25. The first, Fig. 24, shows the alloy of Cu, 88; Sn, 10; Zn, 2 (× 45). It is composed of composite dendrites, with dark etching cores, set in a small amount of a harder material, the eutectoid. The structure can be explained by reference to Fig. 4. Neglecting Zn for a moment, an alloy of, say, Cu, 90; Sn, 10, on cooling down to the temperature of the liquidus curve, AB, begins to crystallize at about 1,000 C., or x. The composition of the first solid to form is given by the solidus curve, Aa, and is y. Now, as freezing continues and the temperature falls, the composition of the liquid follows the curve AB, the composition of the solid follows the curve Aa, until, at 798° C., the last trace of liquid of composition B has disappeared, and the solid has now the composition a, or Sn, 10 per cent. This state of affairs only occurs with slow cooling, such as to allow the composition of the solid to change progressively along the curve Aa by diffusion. Under ordinary conditions, however, cooling is not slow enough to maintain a state of equilibrium, and the diffusion does not keep pace with freezing, so at the temperature of the line abB the solid contains less Sn than a, and some liquid B remains. In other words, the dendrites of the solid, or I, are richer in Cu in the center, or cores, and etch differentially, as seen in Fig. 24. At 795° the reaction takes place, solid a + liquid B = solid b, and all of the liquid is used up. Hence, below 795° C. are dendrites of a, surrounded by a small amount of β, or $I + II$. Then, at 500° C., the β splits up into the eutectoid, as already described. Hence, an alloy, when cold, consists of unhomogeneous dendrites of a set in a small amount of the eutectoid, as in Fig. 24. In regard to the Zn, 2 per cent., Cu, on freezing, will retain Zn to 30 per cent. in solid solution, or 38 per cent. at ordinary temperatures. In the ternary Cu-Zn-Sn diagram a triangle bounded by Cu to Cu, 90-Sn, 10; and Cu to Cu, 64-Zn, 36, gives the approximate limits of the solid solution *a*. In the alloys rich in Sn, the excess is the eutectoid, as in Fig. 24, all the Zn being in solid solution in the dendrites of alpha. In alloys rich in Zn, as tobin bronze, naval brass, manganese-bronze, the excess is the beta Cu-Zn solid solution, the Sn being all in solution, presumably in the alpha, and therefore the resulting structure is that of muntz metal.

1. *Phosphor-Bronze.*

These alloys may be divided into low and high phosphorus. In low-phosphorus alloys the phosphorus is completely dissolved in the alpha. In high-phosphorus alloys the compound Cu_3P is present as a "pseudo-eutectic," mixed with the eutectoid of alpha and Cu_4Sn. The formation is similar to that shown in the area *AnOf* in Fig. 3, taking *r* as the Cu-Cu_3P eutectic at 8.27 per cent., and 700° C., according to Heyn and Bauer,[35] and *f*, the reaction-point *B* of Fig. 4, at Sn, 25 per cent., and 795° C. The point *O* is probably 620° C., and Cu, 81; Sn, 14.2; P, 4.8 per cent., from Hudson and Law's work[36] on phosphor-bronze. Thus, in freezing, the first constituent to form is alpha, as in Fig. 24. On reaching the curve *fO* occurs the reaction which forms beta. At *O* the whole mass goes solid by the reaction: alpha + liquid $O =$ beta + Cu_3P. Then, at 500° C., the beta splits up into the eutectoid, as in ordinary bronzes. Hence, the structure is composed of alpha dendrites in a ground-mass with a eutectic-like structure, a mixture of Cu_3P, Cu_4Sn, and alpha.

2. *Plastic Bronze.*

The structure of the bronzes rich in Pb is typically shown in Fig. 25: Cu, 62; Sn, 11; Pb, 27 (× 65). The black patches are Pb, the background is bronze (alpha + eutectoid). This structure is due to the fact that Pb has only a limited solubility in Cu and in bronze.

In the Cu-Pb alloys, Cu freezes out and enriches the liquid in Pb until at 950° C. it is saturated with Pb, 35 per cent. Further freezing out of Cu causes the precipitation of a second liquid rich in Pb, which freezes independently when the temperature falls after the first liquid has all frozen. This precipitation of the second liquid rich in Pb gives rise to the structure given in Fig. 25, the final freezing taking place at 327° C.

The ternary alloys Cu-Sn-Pb have been studied by Giolitti and Marantonio[37] and by Guertler,[38] who found that the saturation-point of Cu, 65; Pb, 35, at 950° C., was reduced by the addition of Sn both in concentration and temperature. Thus Sn, 10 per cent., reduces the point to 885° C. and Pb 23 per cent., whereas Sn, 20 per cent., reduces it to 796° C. and Pb 10 per cent. Again, the addition of Pb to a Cu-Sn alloy (say Sn, 10; Cu, 90) reduces the freezing-point. Cu-Sn alpha begins to freeze out and enriches the liquid in Pb. At the saturation-point the liquid rich in Pb is precipitated as before; while the transition-point, $\alpha + \text{liquid} = \beta$, is lowered by the addition of Pb, the eutectoid change at 500° C., where the β decomposes, is not lowered thereby, because the composition of the β is not altered by the lead.

In general, then, the structure of ordinary bronze consists of the Cu-Sn solid solution alpha with a small amount of the hard eutectoid filling in the interstitial spaces between the dendrites and grains. Phosphor-bronze is similar, but in addition to the eutectoid Cu_3P is present, with a proportionate lowering of both freezing- and melting-points. Lastly, the plastic bronzes show, in addition to the Cu-Sn solid solution alpha and the euctectoid, more or less patches of Pb, which has reduced the freezing-point and retarded the end of solidification to 327° C., the freezing-point of Pb. The presence of this Pb gives the plastic property.

APPENDIX.

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The Action of Various Commercial Carbonizing-Materials.

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THE practice of carbonizing steel for the purpose of case-hardening has assumed great commercial importance within the past 10 years.

Formerly, case-hardened steel was held in more or less contempt, since it was considered a cheap substitute for tool-steel. This is no longer the case. The development of the motor-car industry, and coincidentally that of modern alloy-steels, has resulted in overcoming many difficulties formerly experienced in case-hardening. This result has been accomplished by increased knowledge regarding the analysis of the steel which would respond most readily to case-hardening, and by more careful methods of treatment of the carbonized steel.

The ordinary method of carbonizing consists in packing the steel with the carbonizing-material in cast-iron boxes and placing them in a furnace at a temperature of from 1,500° to 1,900° F. for a time sufficient to give the required depth of "case." The steel is then either quenched directly from the box, or is allowed to cool without unpacking, and finally given a single hardening-heat, or, for special work, two or three heats.

During the year 1911, more than 100,000 tons of carbonizing-material was sold in the United States, at an average price of probably $55 per ton. More than 85 per cent. of this material consisted of granulated bone.

Within the past four years, many manufactured compounds have been placed on the market, consisting largely of some form of carbon or carbonaceous material, with or without the addition of chemicals.

Steel will absorb carbon placed in contact with it at the temperature of the atmosphere. The reaction is exceedingly slow, but increases rapidly with increasing temperature. Below

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1,500° F., the absorption is too slow to be commercially important. Various causes combine to make it impracticable to carbonize above 1,900° F. In general, carbonizing is carried out at temperatures ranging from 1,550° to 1,750°, and higher temperatures are used only when the quality is not as important as the cost. Broadly speaking, the higher temperatures can be employed for high-grade work when proper facilities are at hand for a careful regulation of the temperature, and a knowledge is possessed of the correct subsequent heat-treatment of the carbonized product.

When steel is carbonized, the carbon does not penetrate in a gradually decreasing content from a high-carbon exterior to the uncarbonized core, but rather in a series of steps; for example, there is usually present a considerable zone of the eutectoid composition, which in the present paper has been assumed to contain 0.90 per cent. of carbon.

Many of the commercial materials behave differently in carbonizing, and even under exactly the same conditions of time and temperature different depths of penetration or different per cents. of carbon, or both, are obtained.

In order to produce good results in case-hardening, a uniform material must be used, and the treatment of the steel subsequent to carbonizing must be suited to the nature of the case produced.

The present investigation was undertaken to compare most of the important commercial carbonizers as to cost of carbonizing, rapidity of carbonizing, and the nature of the resultant carbon-zones.

The method of procedure usually followed in investigating carbonizing-compounds consists in carbonizing small bars for different lengths of time, and then turning off successive layers, usually 0.010 in. deep, and analyzing the turnings for carbon. This method is not entirely satisfactory, as two bars showing practically the same carbon-content might be carbonized in entirely different ways, so that the same subsequent heat-treatment would produce decidedly different results.

Method of Investigation.

The method used in the investigation is the result of more than four years' experience in studying carbonizing and carbonizing-compounds. Twenty-five materials were investigated, designated by the letters *A* to *Y*. For obvious reasons, the names of the compounds are withheld.

Materials *C*, *D*, *I*, and *U* are pure bone obtained from different firms. *U* and *I* consist of bone in which the glue has been partly or wholly extracted. *A* and *K* are bone mixed with other carbonaceous material. *E* consists of partly-roasted nuts; *W* consists largely of husks and kernels of seeds or beans. *X* and *Y* consist of a charred carbonaceous material heavily charged with chemicals.

The remainder of the compounds consist of carbonaceous material in the form of graphite, coal, coke, and charcoal, most of which are charged with various chemicals.

(1) Eight bars of steel, all from the same heat, were selected after check-analysis showed them to be of uniform composition. The analysis of this material was: C, 0.205; P, 0.015; S, 0.025; Mn, 0.53; Si, 0.04 per cent.

These bars, 0.75 in. in diameter, were cut in 0.75-in. lengths. The pieces, each weighed carefully on an analytical balance, were placed in small white cast-iron pots, surrounded by carbonizing-material, as shown in Fig. 1. The cover of the pot was luted with fire-clay. The pots were new, and the same pot was used throughout with the material used in it at the start. Identical tests were carried out on each material at three different temperatures, 1,600°, 1,750°, and 1,900° F., respectively.

Ten pots of each material were packed for each temperature, and the 250 pots were placed on the hearth of a gas-fired furnace, having a hearth of such a width and length that the entire bank of pots was 2 ft. from either end and 8 in. from either side. The pots were arranged in 10 rows of 25 to a row, each row contained one pot of each material, and these separate rows were placed upon a strip of sheet-iron, so that the entire row could be drawn out at the end of each hour. Each strip containing 25 pots was first placed on a large plate of sheet-iron, and the furnace was brought up to heat and kept 25° F. above the chosen temperature for 2 hr. before the heat was placed in it. The furnace was provided with a door at each end, and at the proper time the large plate containing the 250 pots was drawn into the furnace. The furnace-temperature was then maintained constant for 10 hr. Two pyrometer-couples, used for measuring the temperature, were connected to a recording-instrument, which gave a record of each couple every 40 sec. At the end of an hour, the first row of pots was drawn from the furnace. The operation of drawing this strip of sheet-iron upon which the pots were placed occupied about 30 sec. At the end of 2 hr., the next row was drawn, etc. The mass of the brick-work in the furnace used was sufficiently great to prevent sudden changes of temperature. After the pots were cool, the pieces of steel were taken from them, carefully washed and reweighed. They were then sawn in two, polished, etched, and examined under the microscope.

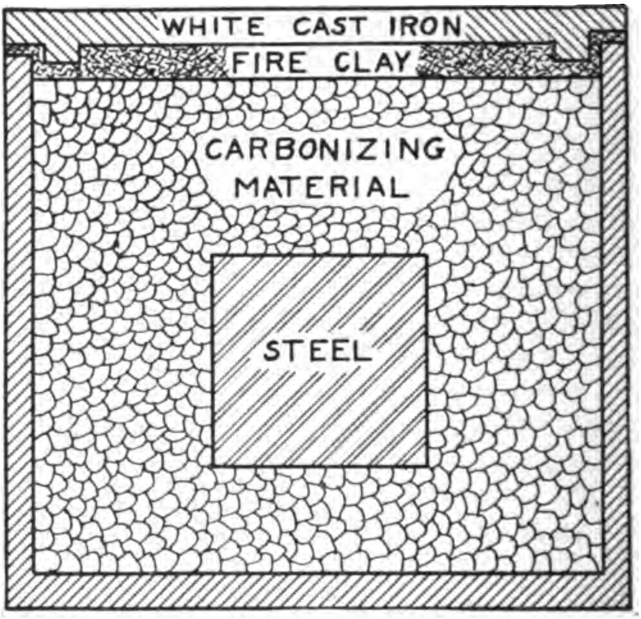


Fig. 1.—A Carbonizing Pot. Cross-Section.

By the use of a micrometer eye-piece, the carbonized surface was measured. First, the depth of the zone of carbon containing more than the eutectic ratio (which was considered as 0.90 per cent.) was measured, next the zone containing the eutectic ratio. The remainder of the carbonization was then divided into two zones, the line of division being estimated as the point which contained 0.50 per cent. of carbon. The percentage of carbon in the extreme outer surface was also estimated. The results of these measurements, weights, and estimations are given in Tables VII. to XXXI.

Materials *X* and *Y* caused a decided pitting of the surface of the steel, and in many cases it was impossible to weigh the pieces accurately.

The results regarding the depths of carbonization and increase in weight, shown graphically in the diagrams of Tables VII. to XXXI., are designated by the letter corresponding to the material, and by the temperature of carbonization; thus *A*–1,600 represents graphically the depths of the various zones of carbon produced by material *A* at a temperature of 1,600° for different lengths of time from 1 to 10 hr. The abscissa, or base-line, represents the surface of the steel. The ordinates erected to this at each hour represent the depth of carbonization. The hypereutectic zone of carbon is distinguished by single cross-hatching; the eutectic zone, by double cross-hatching. The two zones of hypoeutectic carbon are not cross-hatched, nor are the zones of decarbonization. (See *C*–1,750.) The broken line represents the increase in weight.

Repeated Use of Material.

An attempt was made to determine the action of the different materials under repeated use. For this purpose, the material used for 9 and 10 hr. at 1,750° was saved and repacked in two more pots, two being repacked because the shrinkage would finally require more material than the single pot could hold. This second heat was then run for 10 hr. at 1,750°, repacked, and run for two additional heats of 10 hr. This test, of which the results are given in Tables VII. to XXXI., was not entirely satisfactory, however, and is now being repeated on a much larger scale.

The repeated use of material is open to serious objection when uniform results are desired. The ordinary method of mixing part old and part new material is objectionable for the reason that in most places where a great variety of carbonizing is done, the old material is not uniform, since it has been used at different temperatures and for different lengths of time.

The efficiencies considered in various ways, under one use only, and for the three temperatures, of the different materials, are tabulated in Tables I. to VI.

TABLE I.—*Relative Cost of Depositing the Same Weight of Carbon with Different Materials at Three Different Temperatures.*

1,600° F.		1,750° F.		1,900° F.	
Material.	Cost.	Material.	Cost.	Material.	Cost.
V	$1.00	*V*	$1.00	*V*	$1.00
R	1.30	*R*	1.28	*Q*	1.21
M	1.99	*Q*	1.54	*R*	1.38
Q	3.00	*M*	1.73	*M*	1.81
E	4.00	*L*	3.26	*S*	3.43
H	4.54	*S*	3.97	*L*	3.51
J	4.55	*W*	4.24	*W*	3.89
B	4.71	*E*	4.30	*E*	3.89
A	4.85	*H*	4.31	*H*	4.47
I	4.92	*J*	4.60	*P*	4.52
U	5.18	*Y*	4.62	*J*	4.60
P	5.30	*A*	4.70	*A*	5.22
D	5.30	*P*	4.90	*B*	5.22
C	5.84	*B*	5.30	*N*	5.40
F	7.11	*I*	5.40	*O*	5.43
N	7.18	*U*	5.83	*Y*	6.60
S	8.00	*O*	6.37	*I*	6.64
O	8.10	*D*	6.44	*X*	6.78
K	11.71	*N*	6.54	*U*	7.07
....	....	*F*	7.06	*F*	7.20
....	....	*C*	7.33	*D*	7.65
....	....	*K*	10.60	*C*	9.43
....	....	*T*	34.10	*K*	9.65
....	....	....	....	*T*	32.40

These efficiencies were calculated in the following manner: The total increase in weight of the 10 pieces for each temperature was divided by the total number of hours of carbonizing, namely, 55. This gave the average increase in weight for that temperature per hour. (This is a fair average, for in most plants material will be carbonized for different purposes for lengths of time varying from 1 to 10 hr.) This average increase in weight was then used as a basis of comparison in three different ways, which will be explained. In the same way, the total depth of carbonization for each material under each temperature was divided by 55, thus giving an average depth of penetration per hour.

From the commercial standpoint, the material which will deposit the most carbon in the steel at a given cost is the most efficient. Table I. has been calculated upon this basis. Material *V*, being the most efficient (for all three temperatures), was given the value $1. The table is then interpreted thus: by using material *R* (at 1,600°), it will cost $1.30 to deposit the

TABLE II.—*Relative Cost of Carbonizing to a Given Depth with Different Materials at Three Different Temperatures.*

1,600° F.		1,750° F.		1,900° F.	
Material.	Cost.	Material.	Cost.	Material.	Cost.
V	$1.00	*V*	$1.00	*Q*	$1.00
R	1.41	*Q*	1.07	*V*	1.06
M	2.04	*R*	1.48	*R*	1.54
Q	2.43	*M*	2.07	*M*	2.16
H	4.36	*S*	2.97	*S*	3.03
P	4.57	*L*	3.12	*L*	3.60
E	4.65	*W*	3.76	*W*	3.95
I	4.85	*P*	4.61	*P*	4.86
B	4.95	*H*	4.90	*H*	5.07
A	5.02	*J*	5.18	*E*	5.60
J	5.15	*E*	5.35	*J*	5.72
U	5.19	*X*	5.37	*O*	5.86
X	5.20	*Y*	5.52	*X*	6.12
Y	5.20	*A*	5.52	*B*	6.17
D	5.55	*I*	5.57	*Y*	6.19
S	5.73	*U*	5.63	*A*	6.25
C	6.15	*B*	5.76	*I*	6.26
O	6.33	*O*	6.48	*U*	6.27
N	7.88	*D*	6.54	*D*	7.61
F	8.88	*C*	7.00	*N*	7.80
K	10.10	*N*	7.77	*C*	8.61
....	....	*F*	9.05	*K*	9.88
....	....	*K*	9.87	*F*	9.93
....	....	*T*	34.75	*T*	44.75

same weight of carbon as could be deposited by material *V* at a cost of $1. Likewise, material *K* will cost $11.71 under the same conditions.

Table II. is determined in the same manner, with the exception that instead of being calculated upon a basis of efficiency in depositing a given weight of carbon, a given depth of penetration was used.

Table III. gives the relative efficiency of the same weight of different materials in depositing a given weight of carbon; that is, in a determination of the average weight of carbon deposited by the same weight of different materials, that material is the most efficient which will deposit the most carbon. Material *R* was found to be the most efficient in this respect, and was given the value of 100 per cent. *M* was second, with the efficiency of 81.6 per cent. at 1,600° F. As a simple illustration, this can be interpreted as follows: 1 lb. of material *M* will deposit 81.6 per cent. as much carbon as 1 lb. of material *R*.

TABLE III.—*Relative Efficiency of the Same Weight of Different Materials in Depositing a Given Weight of Carbon at Three Different Temperatures.*

1,600° F.		1,750° F.		1,900° F.	
Material.	Efficiency.	Material.	Efficiency.	Material.	Efficiency.
	Per Cent.		Per Cent.		Per Cent.
R	100.0	*R*	100.0	*R*	100.0
M	81.6	*M*	92.3	*M*	95.8
V	81.2	*V*	79.8	*V*	86.5
J	66.4	*Y*	69.1	*J*	69.8
E	57.0	*L*	68.6	*L*	69.0
B	44.8	*J*	64.5	*E*	62.2
F	41.2	*E*	52.0	*N*	57.5
N	40.6	*N*	44.0	*Q*	57.2
A	38.8	*B*	43.2	*X*	53.7
D	31.8	*Q*	41.3	*Y*	52.4
I	31.4	*F*	40.7	*F*	43.2
U	31.3	*A*	39.4	*B*	43.1
P	30.6	*T*	33.7	*S*	40.5
C	28.9	*P*	32.6	*A*	38.4
H	26.3	*S*	32.4	*T*	38.3
Q	21.6	*W*	30.2	*P*	38.3
O	20.0	*H*	29.6	*W*	35.6
K	16.6	*I*	28.0	*O*	31.8
S	16.2	*U*	27.4	*H*	31.0
G	12.2	*D*	25.8	*I*	24.8
....	....	*O*	25.1	*U*	24.5
....	....	*C*	22.6	*D*	23.5
....	....	*K*	18.2	*K*	21.5
....	....	*G*	13.9	*C*	19.1
....	....	....	....	*G*	16.4

Table IV. is calculated the same way as Table III., with the exception that its efficiency is based upon the depth of carbonization given by the same weight of different materials.

Table V. is based upon the relative efficiencies of the same volume of different materials in depositing a given weight of carbon. In this case, material *F* is the most efficient at 1,600° and 1,750° F.; while material *N* is the most efficient at 1,900° F. This table of efficiencies does not take into account the specific gravity of the different materials, and, therefore, other things being equal, the heaviest material will appear as the most efficient.

Table VI. is similar to Table V., except that the depth of carbonization is the basis and not the weight.

Discussion of Results.

A critical discussion of the results of efficiencies given in Tables I. to VI. could be extended indefinitely. We will, there-

TABLE IV.—*Relative Efficiency of the Same Weight of Different Materials in Carbonizing to a Given Depth at Three Different Temperatures.*

1,600° F.		1,750° F.		1,900° F.	
Material.	Efficiency.	Material.	Efficiency.	Material.	Efficiency.
	Per Cent.		Per Cent.		Per Cent.
R	100.0	*R*	100.0	*R*	100.0
V	88.0	*V*	92.5	*V*	91.0
M	86.3	*M*	89.9	*M*	89.5
X	71.1	*L*	83.3	*Q*	77.2
Y	67.8	*X*	72.5	*L*	75.0
J	63.5	*Q*	69.1	*X*	66.2
E	52.8	*Y*	67.2	*J*	62.8
B	46.3	*J*	66.4	*Y*	62.3
A	40.7	*S*	50.0	*S*	50.9
N	40.2	*E*	48.5	*E*	48.3
P	38.4	*N*	42.8	*N*	44.4
F	35.3	*B*	41.7	*B*	40.6
I	34.5	*P*	40.3	*P*	39.8
U	33.9	*W*	39.3	*W*	39.1
D	32.9	*A*	39.0	*A*	35.9
H	32.3	*T*	38.4	*F*	35.0
C	29.8	*F*	36.9	*O*	32.9
Q	28.9	*U*	32.9	*T*	31.1
O	27.8	*I*	31.6	*U*	30.8
S	24.5	*H*	30.3	*H*	30.5
K	20.9	*D*	29.4	*I*	29.3
G	11.0	*O*	28.6	*D*	26.4
....	....	*C*	27.6	*K*	23.5
....	....	*K*	22.6	*C*	23.3
....	....	*G*	19.3	*G*	22.8

fore, consider but a few examples to illustrate the method of making practical use of the data.

The use to which case-hardened steel is to be put should be the deciding factor in selecting the material with which it is to be carbonized, and also to a less extent the temperature of carbonization should be determined from similar considerations.

Assume that a plant was making brake-shoes which required tremendous resistance to wear and but slight resistance to shock. The pressure upon the brake-shoe would likewise be distributed over a considerable area and would not be concentrated in a point or on a line, with the resulting tendency to bend the carbonized surface in a manner similar to the bending of a beam by a load at the center. Under this condition, the main requirements should be a surface containing the maximum amount of carbon possible, since under the proper subse-

TABLE V.—*Relative Efficiency of the Same Volume of Different Materials in Depositing a Given Weight of Carbon at Three Different Temperatures.*

1,600° F.		1,750° F.		1,900° F.	
Material.	Efficiency.	Material.	Efficiency.	Material.	Efficiency.
	Per Cent.		Per Cent.		Per Cent.
F	100.0	*F*	100.0	*N*	100.0
E	99.8	*E*	92.9	*E*	96.0
R	89.0	*Y*	92.5	*F*	92.3
B	88.0	*R*	90.1	*J*	81.0
J	87.7	*N*	87.5	*T*	79.8
C	86.2	*M*	87.0	*R*	78.5
A	84.5	*A*	86.9	*M*	77.7
V	82.4	*J*	86.4	*V*	77.1
D	82.3	*B*	86.2	*B*	74.5
U	81.9	*V*	81.8	*A*	73.6
N	79.9	*T*	80.7	*O*	69.8
I	78.0	*U*	72.4	*W*	66.4
M	75.2	*I*	70.7	*H*	65.3
H	63.0	*C*	68.3	*P*	65.2
P	59.2	*D*	67.2	*Y*	61.0
K	50.5	*H*	65.3	*X*	59.5
O	49.8	*W*	64.8	*K*	57.6
G	30.9	*L*	64.1	*U*	56.2
S	22.4	*P*	64.0	*L*	56.0
Q	19.3	*O*	63.3	*I*	54.2
....	....	*K*	55.7	*D*	53.3
....	....	*S*	45.0	*C*	50.0
....	....	*Q*	37.3	*S*	49.3
....	....	*G*	35.8	*Q*	44.8
....	....	....	....	*G*	36.9

quent treatment this would give the best wearing-surface. Consider, for an example, the relative efficiencies of materials *F* and *K* for this purpose. Assume that a temperature of 1,750° F. was to be used. Both materials would give approximately the same depths of carbon. However, material *K*, excepting for 10 hr., gives a surface carbonization which is not in excess of 0.90 per cent., while *F* gives a surface carbonization of about 1.10 per cent. Material *F* would evidently be far better than *K*. If the surface was to be ground after hardening, most of the high carbon would be removed from *F*, leaving the surface-carbon considerable lower. However, this same grinding would remove most of the 0.90 per cent. of carbon from the steel carbonized with material *K*, and in this case also material *F* would be the better.

Assume that a gear was to be case-hardened. We might

TABLE VI.—*Relative Efficiency of the Same Volume of Different Materials in Carbonizing to a Given Depth at Three Different Temperatures.*

1,600° F.		1,750° F.		1,900° F.	
Material.	Efficiency.	Material.	Efficiency.	Material.	Efficiency.
	Per Cent.		Per Cent.		Per Cent.
E	100.0	*V*	100.0	*V*	100.0
B	98.1	*X*	97.0	*R*	96.4
C	96.8	*T*	96.6	*N*	94.6
Y	96.7	*F*	95.5	*F*	92.0
X	96.7	*R*	95.0	*E*	91.6
V	96.3	*Y*	94.5	*X*	90.5
R	96.2	*J*	92.5	*J*	89.8
A	95.8	*U*	91.5	*W*	89.8
U	95.5	*A*	90.4	*M*	89.3
F	93.5	*E*	90.4	*Y*	89.1
I	92.6	*N*	89.6	*O*	88.6
D	92.0	*W*	88.6	*U*	87.1
J	90.5	*C*	88.1	*B*	86.5
M	85.9	*M*	88.1	*A*	84.6
N	85.2	*B*	87.3	*P*	83.3
H	83.6	*I*	83.6	*T*	79.6
P	80.4	*P*	82.8	*H*	79.2
O	74.6	*L*	81.5	*I*	79.0
K	68.7	*D*	80.6	*K*	77.4
S	36.6	*H*	77.2	*S*	76.2
G	30.3	*O*	75.6	*C*	76.0
Q	27.8	*S*	73.5	*L*	74.8
....	....	*K*	73.0	*Q*	74.4
....	....	*Q*	65.6	*D*	73.8
....	....	*G*	52.4	*G*	62.8

here consider two possible cases: First, that the work was being done by a firm which had very slight knowledge of the heat-treatment necessary to put the gear in the best condition after carbonization; and, second, that the work was being done by a firm which had the necessary knowledge. Suppose material *V* was used by both firms and at a temperature of 1,750° F. After the gear was carbonized and allowed to cool in the carbonizing-pot, the surface would consist of hypereutectic steel; that is, it would contain spines or a net-work of brittle cementite or iron carbide surrounding the cells of pearlite, which contained the eutectic ratio of carbon, *i. e.*, 0.90 per cent. The firm without the necessary heat-treatment knowledge would reheat this gear to the hardening-temperature, which they would find to be slightly less than 1,400° F. This would leave the spines or net-work of cementite unchanged, and as a result the surface would be traversed by this extremely brittle material, which, if the carbonization were not deep enough, might allow a dangerous deflection, due to the concentrated load which must be carried by the carbonized surface, acting in a manner similar to a beam. This would cause minute fractures in the surface, which would finally result in its being pitted and chipped. If, however, the gear was correctly heat-treated, as assumed in the second case, this cementite would be broken up and distributed throughout the surface in minute particles, which would have no embrittleing effect upon it, but would act as particles of extremely hard, wear-resisting material.

In the case above assumed, it would be better that a material similar to *K* or *P* be used in the first case, since, by the use of this material, the outer zone of brittle cementite would not be formed, and although the surface would not be as hard, due to the absence of free cementite, still the tendency towards brittleness would be eliminated.

It will be noted in Table II. that material *Q* stands near the top as regards the efficiency of cost, being the most efficient of all the materials at 1,900° F.

An examination of Table XXIII. shows that at 1,600° no 0.90 per cent. carbon is produced; therefore, steel carbonized with this material at this temperature could not be hardened sufficiently to give a good wearing-surface. This is practically true of 1,750° also. At 1,900°, however, considerable 0.90 per cent. carbon is produced, and if this high temperature was not objectionable, this material would be efficient if nothing else were considered. However, upon comparing *V*–1,900 to *Q*–1,900, it is seen that *V* will deposit a much greater amount and also a greater depth of carbon than *Q* in the same length of time, and will give also a much deeper zone of eutectic and hypereutectic carbon. It is evident, then, that, considering the cost due to the longer time necessarily required by *Q* to give the same depth of carbon, it would probably be considerably less efficient from a practical standpoint.

The question of carbonizing-material is one which usually is not given the attention which its importance demands.

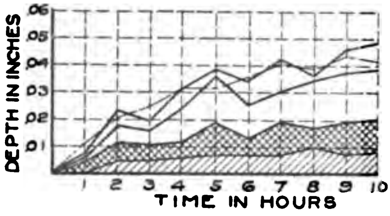
A few sample tests conducted under intelligent direction will often result not only in a great saving of money, but in increased efficiency with regard to the product.

TABLE VII.—*Material "A."*

Hours.	Increase in Weight.	Surface Carbon.	Depth of Carbon.				
			Over 0.90 Per Cent.	0.90 Per Cent.	0.90 to 0.50 Per Cent.	0.50 to 0.20 Per Cent.	Total.
			1,600° F.				
	Grams.	Per Cent.	Inches.	Inches.	Inches.	Inches.	Inches.
1	0.0225	0.90	0.0000	0.0048	0.0012	0.0024	0.0084
2	0.043	0.92	0.0048	0.0072	0.0060	0.0048	0.0228
3	0 0515	0.95	0.0048	0.0060	0.0048	0.0048	0.0204
4	0.064	0.95	0.0060	0.0060	0.0120	0.0072	0.0312
5	0.0645	0.95	0.0072	0.0120	0.0168	0.0024	0.0384
6	0.0745	0.95	0.0072	0.0060	0.0120	0.0084	0.0336
7	0.083	0.98	0.0072	0.0120	0.0108	0.0120	0.0420
8	0.0795	1.00	0.0096	0.0072	0.0168	0.0024	0.0360
9	0.088	0.95	0.0072	0.0120	0.0180	0.0084	0.0456
10	0.0845	1.00	0.0084	0.0120	0.0180	0.0096	0.0480
			1,750° F.				
1	0.0355	0.92	0.0036	0.0036	0.0060	0.0048	0.0180
2	0.0585	1.00	0.0084	0.0096	0.0060	0.0072	0.0312
3	0.072	1.00	0.0072	0.0096	0.0120	0.0048	0.0336
4	0.0955	1.00	0.0156	0.0084	0.0180	0.0084	0.0504
5	0.114	1.05	0.0192	0.0156	0.0144	0.0132	0.0624
6	0.1215	1.10	0.0180	0.0132	0.0240	0.0060	0.0612
7	0.143	1.15	0.0204	0.0180	0.0192	0.0108	0.0684
8	0.1395	0.95	0.0084	0.0300	0.0300	0.0072	0.0756
9	0.148	1.25	0.0204	0.0144	0.0288	0.0060	0.0696
10	0.1475	1.00	0.0144	0.0264	0.0288	0.0180	0.0876
			1,900° F.				
1	0.063	0.90	0.0000	0.0132	0.0120	0.0036	0.0288
2	0.082	0.90	0.0000	0.0144	0.0132	0.0132	0.0408
3	0.059	0.90	....	0.0252	0.0192	0.0144	0.0588
4	0.128	0.90	....	0.0522	0.0348	0.0156	0.0756
5	0.1485	1.05	0.0096	0.0312	0.0252	0.0216	0.0876
6	0.1495	1.05	0.0228	0.0228	0.0252	0.0096	0.0804
7	0.191	0.90	....	0.0492	0.0444	0.0168	0.1104
8	0.2045	1.00	0.0252	0.0564	0.0336	0.0216	0.1368
9	0.223	1.10	0.0216	0.0456	0.0492	0.0192	0.1356
10	0.236	1.00	0.0268	0.0420	0.0588	0.0120	0.1392
			Repeated Use at 1,750° F.				
1st 10	0.1475	1.00	0.0144	0.0264	0.0288	0.0180	0.0876
2d 10	0.1540	1.00	0.0204	0.0192	0.0360	0.0132	0.0888
3d 10	0.1130	0.95	0.0072	0.0168	0.0240	0.0192	0.0672
4th 10	0.097	0.90	....	0.0252	0.0312	0.0132	0.0696

TABLE VIII.—*Material "B."*

Hours.	Increase in Weight.	Surface Carbon.	Depth of Carbon.				
			Over 0.90 Per Cent.	0.90 Per Cent.	0.90 to 0.50 Per Cent.	0.50 to 0.20 Per Cent.	Total.
	Grams.	Per Cent.	Inches.	Inches.	Inches.	Inches.	Inches.
			1,600° F.				
1	0.021	0.90	....	0.0048	0.0024	0.0024	0.0096
2	0.0395	0.90	....	0.0060	0.0060	0.0060	0.0180
3	0.0505	0.93	0.0054	0.0048	0.0120	0.0012	0.0234
4	0.0575	0.95	0.0072	0.0072	0.0096	0.0084	0.0324
5	0.0625	0.90	....	0.0120	0.0180	0.0048	0.0348
6	0.069	1.05	0.0084	0.0036	0.0180	0.0084	0.0384
7	0.074	0.92	0.0048	0.0120	0.0180	0.0060	0.0408
8	0.078	0.93	0.0060	0.0108	0.0180	0.0024	0.0372
9	0.0825	0.93	0.0060	0.0120	0.0180	0.0120	0.0480
10	0.087	0.93	0.0054	0.0204	0.0240	0.0018	0.0516
			1,750° F.				
1	0.035	0.90	....	0.0036	0.0108	0.0096	0.0240
2	0.0545	0.90	....	0.0096	0.0096	0.0084	0.0276
3	0.068	0.95	0.0084	0.0096	0.0168	0.0048	0.0396
4	0.0805	0.95	0.0120	0.0072	0.0132	0.0072	0.0396
5	0.1125	1.15	0.0132	0.0168	0.0144	0.0072	0.0516
6	0.123	1.10	0.0156	0.0168	0.0192	0.0084	0.0600
7	0.135	1.00	0.0192	0.0216	0.0120	0.0072	0.0600
8	0.143	1.10	0.0192	0.0216	0.0288	0.0060	0.0756
9	0.1535	1.15	0.0228	0.0156	0.0252	0.0132	0.0768
10	0.162	1.10	0.0216	0.0240	0.0264	0.0120	0.0840
			1,900° F.				
1	0.058	0.90	....	0.0108	0.0192	0.0084	0.0384
2	0.086	1.00	0.0060	0.0156	0.0168	0.0108	0.0492
3	0.109	0.90	....	0.0300	0.0204	0.0084	0.0588
4	0.124	0.95	0.0048	0.0336	0.0264	0.0168	0.0816
5	0.144	0.93	0.0096	0.0384	0.0228	0.0060	0.0768
6	0.155	1.00	0.0204	0.0252	0.0288	0.0228	0.0972
7	0.185	1.10	0.0108	0.0420	0.0420	0.0168	0.1116
8	0.199	1.10	0.0240	0.0300	0.0504	0.0168	0.1212
9	0.216	1.00	0.0156	0.0480	0.0432	0.0312	0.1380
10	0.228	1.00	0.0228	0.0384	0.0420	0.0372	0.1404
			Repeated Use at 1,750° F.				
1st 10	0.162	1.10	0.0216	0.0240	0.0264	0.0120	0.0840
2d 10	0.120	0.95	0.0108	0.0240	0.0276	0.0192	0.0816
3d 10	0.109	0.90	....	0.0216	0.0324	0.0120	0.0660
4th 10	0.109	0.95	0.0060	0.0204	0.0360	0.0156	0.0780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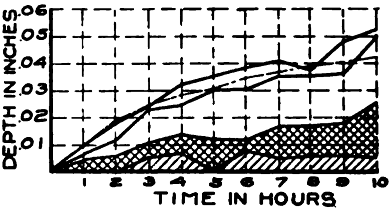
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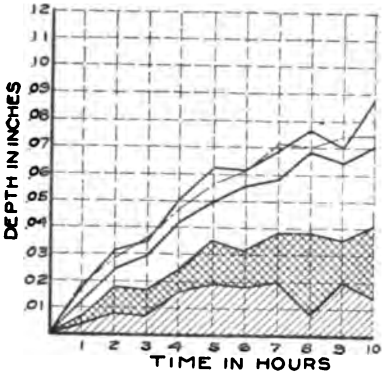
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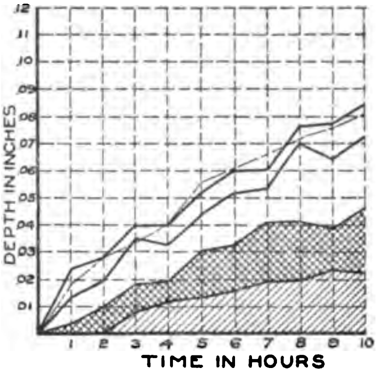
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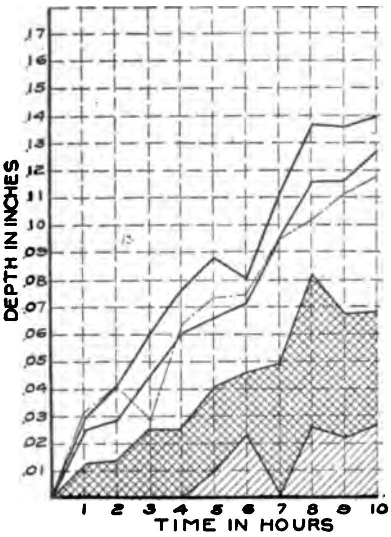
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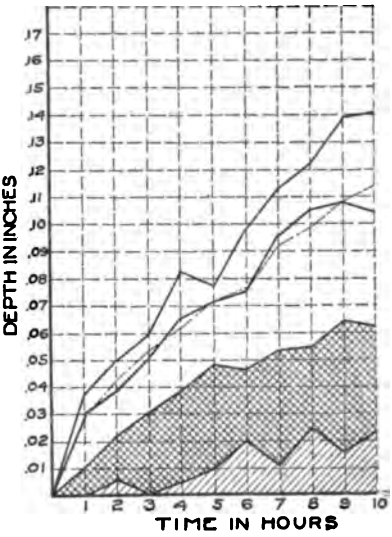

B-1900°

TABLE IX.—*Material "C."*

Hours.	Increase in Weight.	Surface Carbon.	Depth of Carbon.				
			Over 0.90 Per Cent.	0.90 Per Cent.	0.90 to 0.50 Per Cent.	0.50 to 0.20 Per Cent.	Total
1,600° F.							
	Grams.	Per Cent.	Inches.	Inches.	Inches.	Inches.	Inches.
1	0.022	0.90	....	0.0048	0.0036	0.0012	0.0096
2	0.030	0.90	....	0.0048	0.0060	0.0036	0.0144
3	0.0495	0.93	0.0060	0.0048	0.0120	0.0012	0.0240
4	0.058	1.00	0.0072	0.0048	0.0120	0.0048	0.0288
5	0.064	1.00	0.0060	0.0072	0.0144	0.0060	0.0336
6	0.0695	0.95	0.0060	0.0120	0.0180	0.0060	0.0420
7	0.0755	1.00	0.0072	0.0096	0.0180	0.0096	0.0444
8	0.0735	0.95	0.0060	0.0144	0.0120	0.0072	0.0396
9	0.083	1.00	0.0096	0.0048	0.0240	0.0108	0.0492
10	0.0845	0.95	0.0060	0.0144	0.0120	0.0120	0.0444
1,750° F.							
1	0.0385	0.90	....	0.0084	0.0048	0.0048	0.0180
2	0.0595	0.92	0.0072	0.0084	0.0108	0.0024	0.0288
3	0.070	0.95	0.0084	0.0108	0.0156	0.0084	0.0432
4	0.085	1.05	0.0096	0.0120	0.0180	0.0048	0.0444
5	0.1045	1.15	0.0132	0.0204	0.0180	0.0024	0.0540
6	0.115	0.92	0.0084	0.0228	0.0204	0.0180	0.0696
7	0.085	0.80	....	....	0.0552	0.0072	0.0624
8	0.0875	0.00	....	0.0264	0.0324	0.0072	0.0660
9	0.113	0.80	....	....	0.0660	0.0192	0.0852
10	0.087	0.20	....	....	0.0636	0.0084	0.0720
1,900° F.							
1	0.051	0.90	....	0.0108	0.0240	0.0048	0.0396
2	0.072	0.90	....	0.0156	0.0252	0.0144	0.0552
3	0.071	0.95	0.0084	0.0132	0.0264	0.0096	0.0576
4	0.103	0.90	....	0.0252	0.0264	0.0252	0.0768
5	0.1175	0.90	....	0.0228	0.0408	0.0144	0.0780
6	0.113	0.90	....	0.0264	0.0420	0.0276	0.0960
7	0.139	1.00	0.0144	0.0276	0.0288	0.0144	0.0852
8	0.119	0.00	....	0.0408	0.0444	0.0312	0.1164
9	0.099	0.20	....	....	0.0576	0.0240	0.0816
10	0.125	0.00	....	....	0.0852	0.0312	0.1164

Repeated use at 1,750° F.

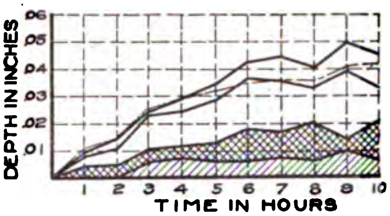
The material would not carbonize after the first ten-hour heat.

TABLE X.—*Material "D."*

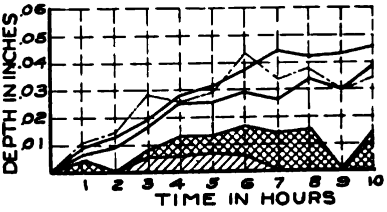
Hours.	Increase in Weight.	Surface Carbon.	Depth of Carbon.				
			Over 0.90 Per Cent.	0.90 Per Cent.	0.90 to 0.50 Per Cent.	0.50 to 0.20 Per Cent.	Total.
				1,600° F.			
	Grams.	Per Cent.	Inches.	Inches.	Inches.	Inches.	Inches.
1	0.023	0.90	....	0.0048	0.0024	0.0024	0.0096
2	0.0295	0.80	....	....	0.0096	0.0056	0.0132
3	0.056	1.00	0.0060	0.0024	0.0084	0.0024	0.0192
4	0.0525	0.95	0.0060	0.0072	0.0120	0.0024	0.0276
5	0.059	0.93	0.0072	0.0060	0.0120	0.0060	0.0312
6	0.0875	0.95	0.0060	0.0108	0.0120	0.0084	0.0372
7	0.067	0.90	....	0.0144	0.0240	0.0060	0.0444
8	0.077	0.90	....	0.0156	0.0180	0.0084	0.0420
9	0.061	0.70	....	....	0.0300	0.0132	0.0432
10	0.068	0.90	....	0.0144	0.0240	0.0072	0.0456
				1,750° F.			
1	0.036	0.90	....	0.0060	0.0060	0.0060	0.0180
2	0.0555	0.95	0.0072	0.0060	0.0096	0.0048	0.0276
3	0.0695	1.10	0.0096	0.0108	0.0096	0.0060	0.0360
4	0.0865	1.15	0.0132	0.0120	0.0168	0 0060	0.0480
5	0.1065	1.10	0.0120	0.0156	0.0168	0.0060	0.0504
6	0.113	1.10	0.0192	0.0168	0.0240	0.0084	0.0684
7	0.1115	0.90	....	0.0300	0.0240	0.0060	0.0600
8	0.099	0.80	....	....	0.0660	0.0072	0.0732
9	0.077	0.00	....	....	0.0540	0.0108	0.0648
10	0.078	0.00	....	....	0.0360	0.0144	0.0504
				1,900° F.			
1	0.051	0.90	....	0.0108	0.0144	0.0072	0.0324
2	0.073	0.90	....	0.0120	0.0264	0.0132	0.0516
3	0.080	0.90	....	0.0264	0.0180	0.0144	0.0588
4	0.112	0.90	....	0.0480	0.0108	0.0156	0.0744
5	0.170	0.90	....	0.0288	0.0408	0.0132	0.0828
6	0.125	0.95	0.0096	0.0240	0.0396	0.0252	0.0984
7	0.090	0.70	....	....	0.0840	0.0204	0.1044
8	0.109	0.20	....	....	0.0648	0.0168	0.0816
9	0.160	0.90	....	0.00576	0.0360	0.0180	0.1116
10	0.105	0.20	....	....	....	0.0840	0.0840
				Repeated Use at 1,750° F.			
2d 10	0.016	0.20	....	....	....	slight	....

This material would not carbonize after this heat.

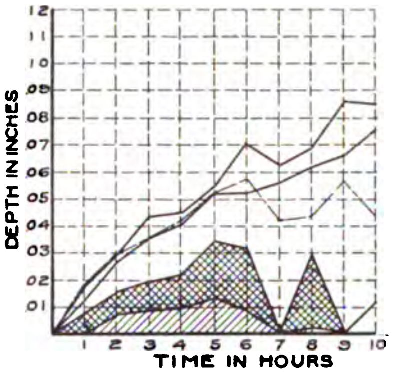
C-1600°



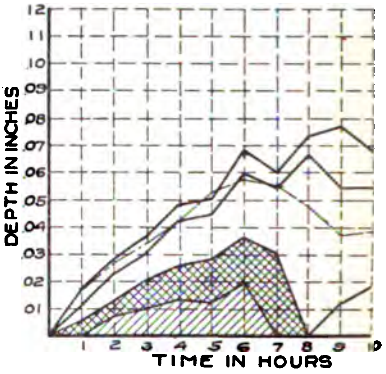
D-1600°



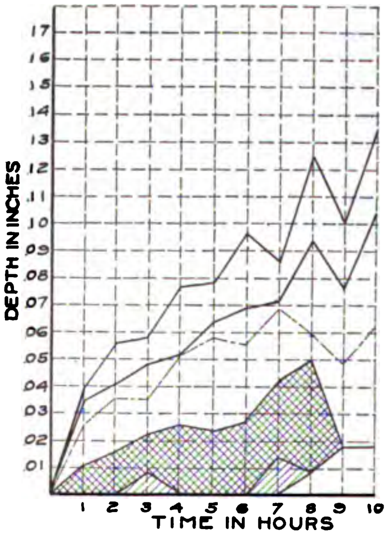
C-1750°



D-1750°



C-1900°



D-1900°

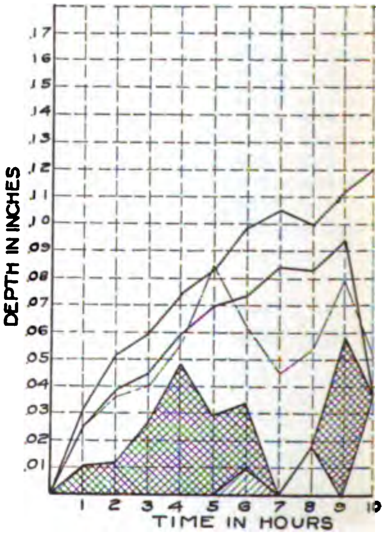


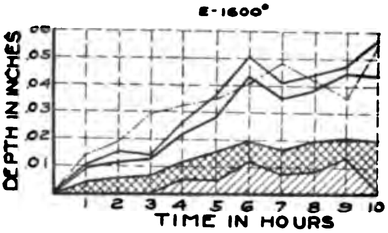
TABLE XI.—*Material "E."*

1,600° F.

Hours.	Increase in Weight.	Surface Carbon.	Depth of Carbon.				
			Over 0.90 Per Cent.	0.90 Per Cent.	0.90 to 0.50 Per Cent.	0.50 to 0.20 Per Cent.	Total.
	Grams.	Per Cent.	Inches.	Inches.	Inches.	Inches.	Inches.
1	0.027	0.90	....	0.0048	0.0048	0.0012	0.0108
2	0.039	0.90	....	0.0060	0.0060	0.0036	0.0156
3	0.059	0.90	....	0.0072	0.0060	0.0012	0.0144
4	0.065	0.93	0.0060	0.0060	0.0096	0.0048	0.0264
5	0.070	0.92	0.0048	0.0108	0.0120	0.0084	0.0360
6	0.082	0.95	0.0120	0.0072	0.0240	0.0072	0.0504
7	0.096	0.95	0.0072	0.0084	0.0192	0.0060	0.0408
8	0.084	1.00	0.0084	0.0108	0.0180	0.0060	0.0432
9	0.072	1.20	0.0132	0.0072	0.0240	0.0024	0.0468
10	0.110	0.90	....	0.0192	0.0240	0.0132	0.0564
1,750° F.							
1	0.042	0.92	0.0048	0.0072	0.0060	0.0060	0.0240
2	0.072	1.05	0.0120	0.0060	0.0072	0.0060	0.0312
3	0.088	1.00	0.0120	0.0096	0.0168	0.0048	0.0432
4	0.098	1.05	0.0144	0.0168	0.0060	0.0048	0.0420
5	0.094	1.10	0.0108	0.0120	0.0108	0.0084	0.0420
6	0.138	1.15	0.0228	0.0168	0.0180	0.0072	0.0648
7	0.144	1.15	0.0216	0.0168	0.0204	0.0096	0.0684
8	0.146	1.15	0.0228	0.0192	0.0216	0.0132	0.0768
9	0.170	1.20	0.0204	0.0300	0.0228	0.0096	0.0828
10	0.158	1.20	0.0276	0.0204	0.0300	0.0048	0.0828
1,900° F.							
1	0.093	1.10	0.0132	0.0132	0.0048	0.0060	0.0372
2	0.124	1.15	0.0180	0.0156	0.0156	0.0108	0.0600
3	0.156	1.10	0.0276	0.0132	0.0240	0.0048	0.0696
4	0.173	1.15	0.0276	0.0240	0.0276	0.0084	0.0876
5	0.198	1.20	0.0300	0.0240	0.0360	0.0096	0.0996
6	0.227	1.25	0.0408	0.0216	0.0288	0.0136	0.1068
7	0.212	1.15	0.0408	0.0252	0.0336	0.0168	0.1164
8	0.211	1.20	0.0372	0.0240	0.0324	0.0072	0.1008
9	0.268	1.25	0.0396	0.0480	0.0420	0.0120	0.1416
10	0.274	1.20	0.0420	0.0444	0.0420	0.0192	0.1476
Repeated Use at 1,750°.							
1st 10	0.158	1.20	0.0276	0.0204	0.0300	0.0048	0.0828
2d 10	0.136	1.10	0.0168	0.0204	0.0216	0.0144	0.0732
3d 10	0.087	0.95	0.0144	0.0168	0.0240	0.0120	0.0672
4th 10	0.121	0.95	0.0108	0.0264	0.0180	0.0096	0.0648

TABLE XII.—*Material "F."*

Hours.	Increase in Weight.	Surface Carbon.	Depth of Carbon.				
			Over 0.90 Per Cent.	0.90 Per Cent.	0.90 to 0.50 Per Cent.	0.50 to 0.20 Per Cent.	Total.
			1,600° F.				
	Grams.	Per Cent.	Inches.	Inches.	Inhes.	Inches.	Inches.
1	0.020	0.90	....	0.0036	0.0048	0.0024	0.0108
2	0.040	0.90	....	0.0072	0.0060	0.0024	0.0156
3	0.050	0.90	....	0.0060	0.0060	0.0048	0.0168
4	0.070	0.90	....	0.0108	0.0168	0.0012	0.0288
5	0.077	1.00	0.0072	0.0024	0.0168	0.0072	0.0336
6	0.082	1.00	0.0066	0.0036	0.0144	0.0096	0.0342
7	0.088	0.92	0.0072	0.0192	0.0060	0.0048	0.0372
8	0.083	1.00	0.0084	0.0060	0.0180	0.0072	0.0396
9	0.088	1.10	0.0084	0.0144	0.0180	0.0036	0.0444
10	0.108	0.93	0.0120	0.0132	0.0240	0.0084	0.0576
			1,750° F.				
1	0.040	1.00	0.0108	0.0120	0.0036	0.0024	0.0288
2	0.070	1.10	0.0084	0.0096	0.0060	0.0072	0.0312
3	0.079	1.10	0.0120	0.0108	0.0120	0.0048	0.0396
4	0.101	1.00	0.0156	0.0108	0.0156	0.0084	0.0504
5	0.101	1.20	0.0168	0.0108	0.0180	0.0084	0.0540
6	0.138	1.05	0.0216	0.0240	0.0144	0.0060	0.0660
7	0.169	1.10	0.0240	0.0180	0.0240	0.0096	0.0756
8	0.166	1.05	0.0252	0.0348	0.0096	0.0036	0.0732
9	0.180	1.05	0.0312	0.0180	0.0240	0.0084	0.0816
10	0.194	1.20	0.0264	0.0264	0.0240	0.0120	0.0888
			1,900° F.				
1	0.076	1.10	0.0084	0.0108	0.0144	0.0072	0.0408
2	0.108	1.05	0.0132	0.0180	0.0204	0.0144	0.0660
3	0.140	1.15	0.0180	0.0192	0.0180	0.0168	0.0720
4	0.166	1.15	0.0240	0.0288	0.0228	0.0168	0.0924
5	0.198	1.15	0.0240	0.0300	0.0300	0.0180	0.1020
6	0.200	1.20	0.0336	0.0288	0.0240	0.0156	0.1020
7	0.212	1.15	0.0384	0.0180	0.0360	0.0108	0.1032
8	0.218	1.15	0.0396	0.0288	0.0288	0.0192	0.1164
9	0.258	1.20	0.0420	0.0420	0.0300	0.0144	0.1284
10	0.286	1.20	0.0612	0.0264	0.0420	0.0192	0.1488
			Repeated Use at 1,750°.				
1st 10	0.194	1.20	0.0264	0.0264	0.0240	0.0120	0.0888
2d 10	0.167	1.10	0.0264	0.0240	0.0300	0.0060	0.0864
3d 10	0.053	1.10	0.0276	0.0180	0.0240	0.0192	0.0888
4th 10	0.063	1.10	0.0288	0.0144	0.0372	0.0132	0.0936


E-1600°
DEPTH IN INCHES
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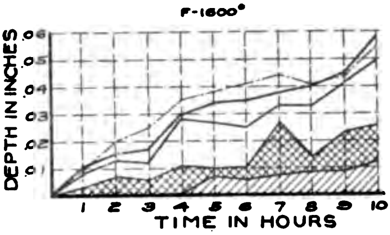
.04
.03
.02
.01
1 2 3 4 5 6 7 8 9 10
TIME IN HOURS


F-1600°
DEPTH IN INCHES
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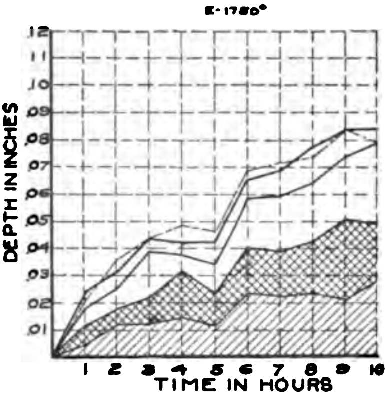
.01
1 2 3 4 5 6 7 8 9 10
TIME IN HOURS


E-1750°
DEPTH IN INCHES
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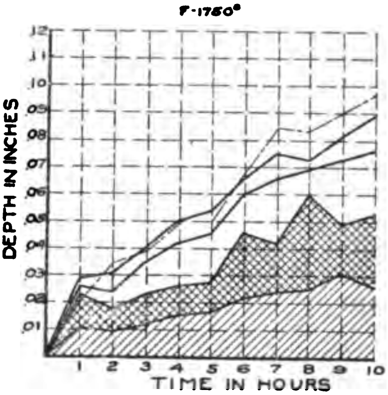
1 2 3 4 5 6 7 8 9 10
TIME IN HOURS


F-1750°
DEPTH IN INCHES
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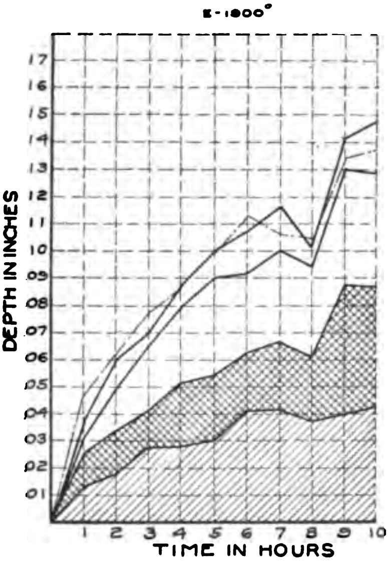
.01
1 2 3 4 5 6 7 8 9 10
TIME IN HOURS


E-1900°
DEPTH IN INCHES
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.16
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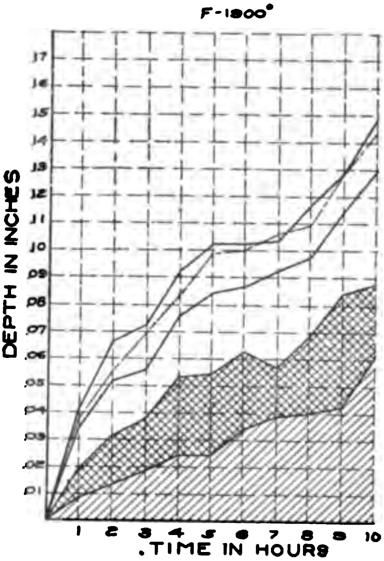
1 2 3 4 5 6 7 8 9 10
TIME IN HOURS


F-1900°
DEPTH IN INCHES
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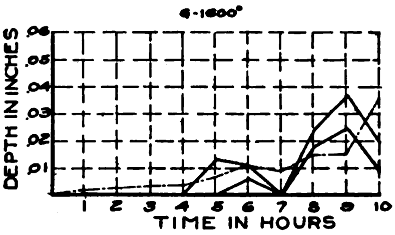
1 2 3 4 5 6 7 8 9 10
TIME IN HOURS

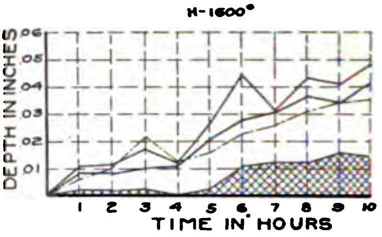
TABLE XIII.—*Material "G."*

Hours.	Increase in Weight.	Surface Carbon.	Depth of Carbon.				
			Over 0.90 Per Cent.	0.90 Per Cent.	0.90 to 0.50 Per Cent.	0.50 to 0.20 Per Cent.	Total.
	Grams.	Per Cent.	Inches.	Inches.	Inches.	Inches.	Inches.
1,600° F.							
1	0.0035	No noticeable increase in carbon.				....	0.0000
2	0.0070	No noticeable increase in carbon.				....	0.0000
3	0.0085	No noticeable increase in carbon.				....	0.0000
4	0.0085	No noticeable increase in carbon.				....	0.0000
5	0.0145	0.30	....	....	....	0.0132	0.0132
6	0.0225	0.80	....	....	0.0060	0.0048	0.0108
7	0.0185	....	....	....	....	....	0.0000
8	0.031	0.70	....	....	0.0180	0.0060	0.0240
9	0.031	0.70	....	....	0.0240	0.0120	0.0360
10	0.073	0.60	....	....	0.0084	0.0108	0.0192
1,750° F.							
1	0.018	0.50	....	....	....	0.0120	0.0120
2	0.0275	0.80	....	....	0.0096	0.0084	0.0180
3	0.0375	0.70	....	....	0.0300	0.0072	0.0372
4	0.041	0.60	....	....	0.0216	0.0084	0.0300
5	0.036	0.70	....	....	0.0132	0.0072	0.0204
6	0.025	0.60	....	....	0.0132	0.0132	0.0264
7	0.064	0.70	....	....	0.0420	0.0072	0.0492
8	0.0605	0.80	....	....	0.0276	0.0084	0.0360
9	0.063	0.70	....	....	0.0360	0.0084	0.0444
10	0.0705	0.70	....	....	0.0420	0.0072	0.0492
1,900° F.							
1	0.022	0.60	....	....	0.0084	0.0096	0.0180
2	0.041	0.70	....	....	0.0168	0.0168	0.0336
3	0.057	0.90	....	0.0084	0.0168	0.0276	0.0528
4	0.071	0.85	....	....	0.0408	0.0072	0.0480
5	0.080	0.90	....	0.0168	0.0528	0.0144	0.0840
6	0.080	0.90	....	0.0048	0.0528	0.0264	0.0840
7	0.085	0.90	....	0.0132	0.0360	0.0288	0.0780
8	0.092	0.85	....	....	0.0576	0.0288	0.0864
9	0.095	0.90	....	0.0084	0.0576	0.0192	0.0852
10	0.121	0.90	....	0.0120	0.0624	0.0192	0.0936
Repeated Use at 1,750°.							
1st 10	0.0705	0.70	....	....	0.0420	0.0072	0.0492
2d 10	0.092	0.90	....	0.0108	0.0408	0.0228	0.0744
3d 10	0.102	0.90	....	0.0300	0.0288	0.0156	0.0744
4th 10	0.089	0.90	....	0.0228	0.0324	0.0072	0.0624

TABLE XIV.—*Material "H."*

Hours.	Increase in Weight.	Surface Carbon.	Depth of Carbon.				
			Over 0.90 Per Cent.	0.90 Per Cent.	0.90 to 0.50 Per Cent.	0.50 to 0.20 Per Cent.	Total.
1,600° F.							
	Grams.	Per Cent.	Inches.	Inches.	Inches.	Inches.	Inches.
1	0.014	0.90	....	0.0024	0.0060	0.0024	0.0108
2	0.021	0.90	....	0.0018	0.0060	0.0036	0.0114
3	0.044	0.90	....	0.0024	0.0072	0.0072	0.0168
4	0.026	0.80	....	....	0.0108	0.0012	0.0120
5	0.035	0.90	....	0.0024	0.0180	0.0060	0.0264
6	0.046	0.90	....	0.0108	0.0168	0.0068	0.0444
7	0.053	0.90	....	0.0120	0.0180	0.0012	0.0312
8	0.062	0.90	....	0.0120	0.0240	0.0072	0.0432
9	0.071	0.90	....	0.0156	0.0180	0.0072	0.0408
10	0.073	0.90	....	0.0144	0.0264	0.0072	0.0480
1,750° F.							
1	0.023	0.90	....	0.0036	0.0072	0.0048	0.0156
2	0.038	0.90	....	0.0084	0.0108	0.0072	0.0264
3	0.057	0.90	....	0.0144	0.0156	0.0084	0.0384
4	0.069	0.90	....	0.0132	0.0168	0.0072	0.0372
5	0.071	0.90	....	0.0156	0.0204	0.0036	0.0396
6	0.078	0.90	....	0.0156	0.0216	0.0072	0.0444
7	0.099	0.90	....	0.0288	0.0216	0.0120	0.0624
8	0.112	1.05	0.0120	0.0288	0.0168	0.0084	0.0660
9	0.126	1.00	0.0180	0.0204	0.0312	0.0048	0.0744
10	0.136	1.10	0.0180	0.0228	0.0240	0.0072	0.0720
1,900° C.							
1	0.042	0.90	....	0 0048	0.0204	0.0036	0 0288
2	0.073	0.90	....	0.0240	0.0180	0.0156	0.0576
3	0.096	0.90	....	0.0252	0.0168	0.0084	0.0504
4	0.116	0.90	....	0.0384	0.0300	0.0072	0.0756
5	0.136	0.93	0.0084	0.0252	0.0336	0.0204	0.0876
6	0.146	1.00	0.0180	0.0264	0.0300	0.0228	0.0972
7	0.159	0.95	0.0192	0.0288	0.0396	0.0156	0.1032
8	0.167	0.95	0.0276	0.0240	0.0360	0.0084	0.0960
9	0.174	0.90	....	0.0444	0.0348	0.0228	0.1020
10	0.209	0.92	0.0132	0.0480	0.0552	0.0216	0.1380
Repeated Use at 1,750°.							
1st 10	0.136	1.10	0.0180	0.0228	0.0240	0.0072	0.0720
2d 10	0.040	0.60	....	....	0.0348	0.0132	0.0480
3d 10	0.092	0.90	....	0.0264	0.0288	0.0132	0.0684
4th 10	0.084	0.90	....	0.0324	0.0240	0.0084	0.0648


G-1600°
DEPTH IN INCHES
.06
.05
.04
.03
.02
.01
1 2 3 4 5 6 7 8 9 10
TIME IN HOURS


H-1600°
DEPTH IN INCHES
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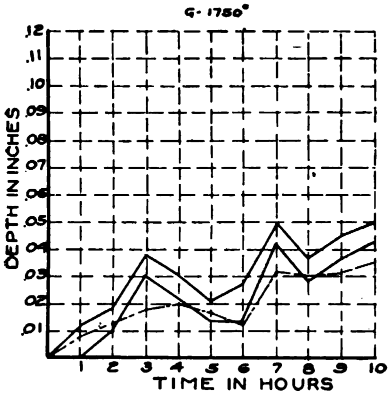
1 2 3 4 5 6 7 8 9 10
TIME IN HOURS


G-1750°
DEPTH IN INCHES
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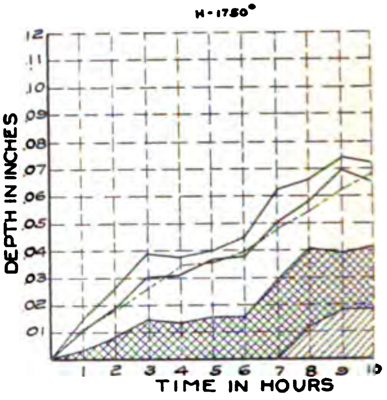
1 2 3 4 5 6 7 8 9 10
TIME IN HOURS


H-1750°
DEPTH IN INCHES
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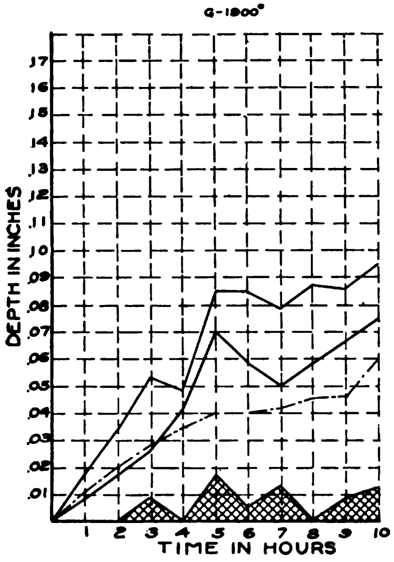
1 2 3 4 5 6 7 8 9 10
TIME IN HOURS


G-1900°
DEPTH IN INCHES
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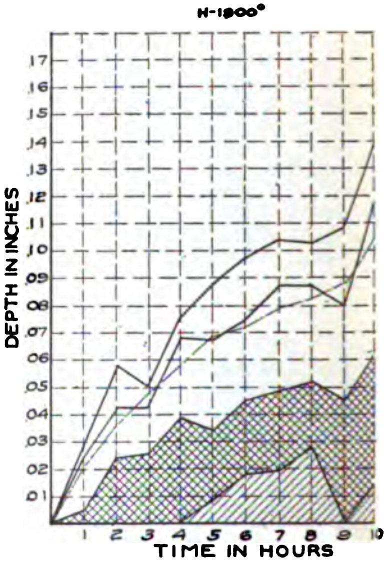
1 2 3 4 5 6 7 8 9 10
TIME IN HOURS


H-1900°
DEPTH IN INCHES
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1 2 3 4 5 6 7 8 9 10
TIME IN HOURS

TABLE XV.—*Material "I."*

Hours.	Increase in Weight.	Surface Carbon.	Depth of Carbon.				
			Over 0.90 Per Cent.	0.90 Per Cent.	0.90 to 0.50 Per Cent.	0 50 to 0.20 Per Cent.	Total.
1,600° F.							
	Grams.	Per Cent.	Inches.	Inches.	Inches.	Inches.	Inches.
1	0.022	0.90	....	0.0036	0.0048	0.0012	0.0096
2	0.036	0.90	....	0.0072	0.0060	0.0036	0.0192
3	0.041	0.90	....	0.0120	0.0060	0.0060	0.0240
4	0.041	0.90	....	0.0096	0.0048	0.0048	0.0192
5	0.047	1.00	0.0024	0.0060	0.0096	0.0048	0.0228
6	0.062	0.90	0.0120	0.0120	0.0180	0.0012	0.0432
7	0.072	0.95	0.0072	0.0096	0.0144	0.0072	0.0384
8	0.077	1.00	0.0108	0.0036	0.0180	0.0036	0.0360
9	0.058	0.75	....	....	0.0444	0.0072	0.0516
10	0.095	1.05	0.0120	0.0108	0.0216	0.0072	0.0516
1,750° F.							
1	0.039	0.90	....	0.0096	0.0060	0.0024	0.0180
2	0.063	1.10	0.0084	0.0084	0.0096	0.0060	0.0324
3	0.078	1.15	0.0096	0.0180	0.0108	0.0072	0.0456
4	0.083	1.10	0.0108	0.0180	0.0096	0.0060	0.0444
5	0.096	1.00	0.0132	0.0156	0.0168	0.0108	0.0564
6	0.096	1.10	0.0084	0.0204	0.0156	0.0084	0.0528
7	0.083	0.92	0.0084	0.0156	0.0240	0.0060	0.0540
8	0.125	1.15	0.0240	0.0180	0.0180	0.0084	0.0684
9	0.130	0.90	....	0.0420	0.0300	0.0120	0.0840
10	0.082	0.00	....	.. ..	0.0480	0.0120	0.0600
1,900° F.							
1	0.053	0.90	....	0.0096	0.0204	0.0072	0.0372
2	0.078	0.90	....	0.0192	0.0216	0.0072	0.0480
3	0.101	0.90	....	0.0180	0.0288	0.0072	0.0540
4	0.118	0.90	....	0.0228	0.0408	0.0192	0.0828
5	0.139	0.95	0.0144	0.0228	0.0360	0.0144	0.0876
6	0.147	0.93	0.0084	0.0324	0.0360	0.0240	0.1008
7	0.155	0.90	....	0.0432	0.0492	0.0060	0.0984
8	0.099	0.20	....	....	0.0900	0.0084	0.0984
9	0.096	0.00	. ...	....	0.0780	0.0252	0.1032
10	0.107	0.20	....	....	0.0948	0.0288	0.1236

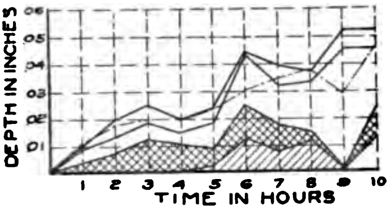
Repeated Use at 1,750° F.

This material would not carbonize after this heat.

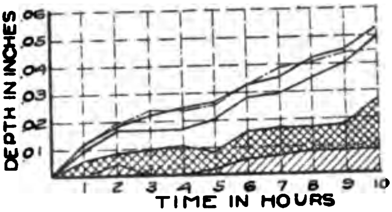
TABLE XVI.—*Material "J."*

Hours.	Increase in Weight.	Surface Carbon.	Depth of Carbon.				
			Over 0.90 Per Cent.	0.90 Per Cent.	0.90 to 0.50 Per Cent.	0.50 to 0.20 Per Cent.	Total.
1,600° F.							
	Grams.	Per Cent.	Inches.	Inches.	Inches.	Inches.	Inches.
1	0.025	0.90	....	0.0060	0.0036	0.0024	0.0120
2	0.038	0.92	0.0048	0.0036	0.0084	0.0012	0.0180
3	0.048	0.90	....	0.0096	0.0072	0.0048	0.0216
4	0.049	0.90	....	0.0108	0.0060	0.0072	0.0240
5	0.054	0.95	0.0024	0.0060	0.0120	0.0060	0.0164
6	0.067	0.95	0.0060	0.0096	0.0120	0.0048	0.0324
7	0.076	0.95	0.0072	0.0096	0.0120	0.0060	0.0348
8	0.083	1.00	0.0084	0.0084	0.0180	0.0072	0.0420
9	0.088	1.00	0.0084	0.0096	0.0216	0.0048	0.0444
10	0.091	1.10	0.0084	0.0180	0.0228	0.0036	0.0528
1,750° F.							
1	0.041	0.90	....	0.0096	0.0084	0.0012	0.0192
2	0.067	1.15	0.0084	0.0072	0.0168	0.0024	0.0348
3	0.085	1.20	0.0120	0.0108	0.0156	0.0060	0.0444
4	0.098	1.15	0.0180	0.0120	0.0120	0.0060	0.0480
5	0.106	1.15	0.0168	0.0108	0.0108	0.0120	0.0504
6	0.105	1.15	0.0168	0.0120	0.0180	0.0120	0.0588
7	0.105	1.00	0.0132	0.0168	0.0240	0.0072	0.0612
8	0.143	1.20	0.0240	0.0240	0.0360	0.0084	0.0924
9	0.157	1.20	0.0228	0.0216	0.0300	0.0084	0.0828
10	0.162	1.05	0.0264	0.0312	0.0108	0.0156	0.0840
1,900° F.							
1	0.067	1.00	0.0072	0.0168	0.0096	0.0096	0.0432
2	0.097	1.00	0.0108	0.0132	0.0276	0.0096	0.0612
3	0.128	1.10	0.0192	0.0216	0.0216	0.0120	0.0744
4	0.160	1.00	0.0156	0.0312	0.0288	0.0204	0.0960
5	0.173	1.00	0.0180	0.0288	0.0324	0.0180	0.0972
6	0.187	1.10	0.0264	0.0276	0.0360	0.0192	0.1092
7	0.198	1.00	0.0192	0.0444	0.0408	0.0084	0.1128
8	0.203	1.20	0.0312	0.0432	0 0264	0.0084	0.1092
9	0.213	1.10	0.0240	0.0312	0.0528	0.0132	0.1212
10	0.210	1.15	0.0276	0.0384	0.0348	0.0228	0.1236
Repeated Use at 1,750°.							
1st 10	0 162	1.05	0.0264	0.0312	0.0108	0.0156	0.0840
2d 10	0.128	1.00	0.0168	0.0180	0.0264	0.0132	0.0744
3d 10	0.120	0.90	....	0.0360	0.0288	0.0108	0.0756
4th 10	0.097	0.90	....	0.0228	0.0384	0.0096	0.0708

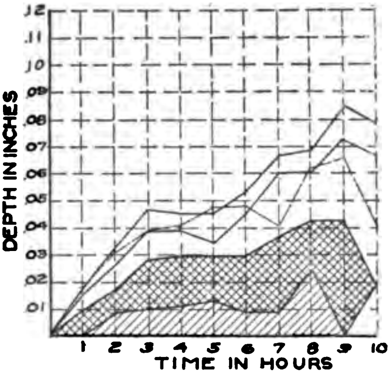
I-1600°



J-1600°



I-1750°



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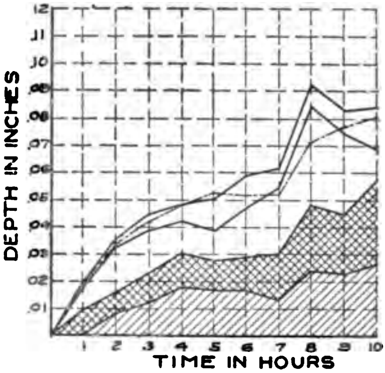
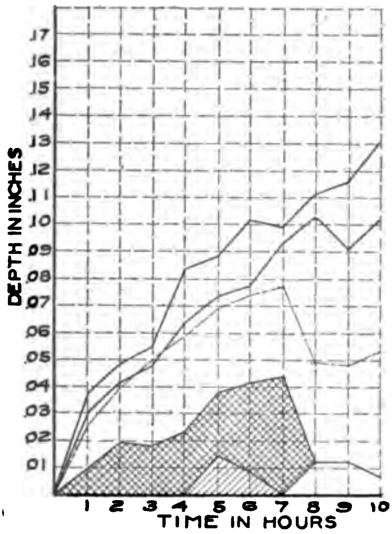


I-1900°



J 1900°

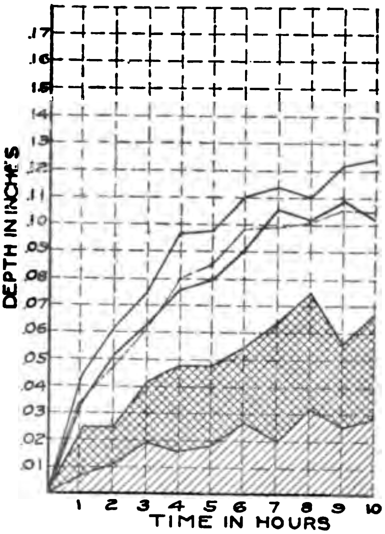


TABLE XVII.—*Material "K."*

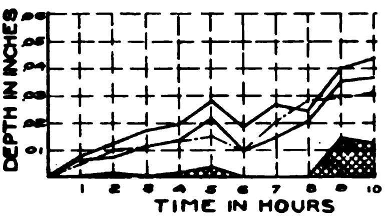
Hours.	Increase in Weight.	Surface Carbon.	Depth of Carbon.				
			Over 0.90 Per Cent.	0.90 Per Cent.	0.90 to 0.50 Per Cent.	0.50 to 0.20 Per Cent.	Total.
	Grams.	Per Cent.	Inches.	Inches.	Inches.	Inches.	Inches.
			1,600° F.				
1	0.0115	0.80	....	....	0.0060	0.0012	0.0072
2	0.0205	0.90	....	0.0012	0.0060	0.0048	0.0120
3	0.0235	0.80	....	....	0.0108	0.0060	0.0168
4	0.026	0.90	....	0.0012	0.0120	0.0060	0.0192
5	0.0325	0.90	....	0.0036	0.0180	0.0060	0.0276
6	0.021	0.60	....	....	0.0096	0.0084	0.0180
7	0.0425	0.70	....	....	0.0144	0.0120	0.0264
8	0.057	0.80	....	....	0.0204	0.0036	0.0240
9	0.0585	0.90	....	0.0144	0.0204	0.0048	0.0396
10	0.0635	0.90	....	0.0120	0.0240	0.0072	0.0432
			1,750° F.				
1	0.023	0.70	....	....	0.0108	0.0024	0.0132
2	0.038	0.90	....	0.0084	0.0120	0.0060	0.0264
3	0.051	0.90	....	0.0084	0.0216	0.0060	0.0360
4	0.0615	0.90	....	0.0120	0.0192	0.0072	0.0384
5	0.067	0.90	....	0.0120	0.0180	0.0144	0.0444
6	0.0705	0.90	....	0.0180	0.0192	0.0072	0.0444
7	0.072	0.90	....	0.0180	0.0288	0.0012	0.0480
8	0.075	0.90	....	0.0192	0.0228	0.0084	0.0504
9	0.109	0.90	....	0.0288	0.0300	0.0132	0.0720
10	0.122	1.00	0.0120	0.0228	0.0360	0.0160	0.0768
			1,900° F.				
1	0.038	0.90	....	0.0072	0.0168	0.0108	0.0348
2	0.063	0.90	....	0.0144	0.0228	0.0192	0.0564
3	0.082	0.90	....	0.0144	0.0324	0.0168	0.0636
4	0.100	0.90	....	0.0192	0.0396	0.0072	0.0660
5	0.120	0.90	....	0.0276	0.0456	0.0084	0.0816
6	0.133	0.90	....	0.0300	0.0408	0.0192	0.0900
7	0.140	0.95	0.0132	0.0288	0.0360	0.0252	0.1032
8	0.157	1.00	0.0168	0.0204	0.0516	0.0108	0.0996
9	0.165	1.00	0.0144	0.0312	0.0408	0.0252	0.1116
10	0.164	0.90	....	0.0480	0.0480	0.0144	0.1104
			Repeated Use at 1,750°.				
1st 10	0.122	1.00	0.0120	0.0228	0.0360	0.0060	0.0768
2d 10	0.133	0.95	0.0084	0.0276	0.0336	0.0144	0.0840
3d 10	0.046	0.90	....	0.0108	0.0300	0.0096	0.0504
4th 10	0.096	0.90	....	0.0228	0.0372	0.0132	0.0732

TABLE XVIII.—*Material "L."*

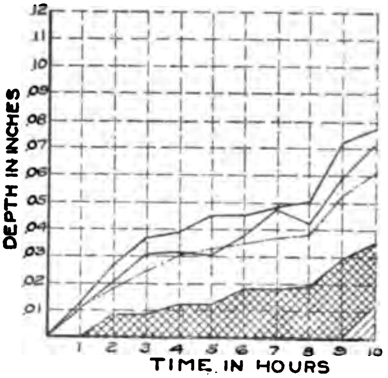
(This material was not used at 1,600°.)

Hours.	Increase in Weight.	Surface Carbon.	Depth of Carbon.				
			Over 0.90 Per Cent.	0.90 Per Cent.	0.90 to 0.50 Per Cent.	0.50 to 0.20 Per Cent.	Total.
	1,750° F.						
	Grams.	Per Cent.	Inches.	Inches.	Inches.	Inches.	Inches.
1	0.036	0.90	....	0.0096	0.0096	0.0012	0.0204
2	0 048	0.90	....	0.0060	0.0120	0.0060	0.0240
3	0.0685	1.00	0.0096	0.0096	0.0144	0.0036	0.0372
4	0.0715	0.90	....	0.0120	0.0216	0.0084	0.0420
5	0.064	0.90	....	0.0132	0.0240	0.0060	0.0432
6	0.0865	0.90	....	0.0276	0.0192	0.0060	0.0528
7	0.089	0.93	0.0060	0.0180	0.0240	0.0108	0.0588
8	0.084	0.95	0.0084	0.0180	0.0224	0.0072	0.0660
9	0.124	1.10	0.0120	0.0300	0.0300	0.0120	0.0840
10	0.1215	0.90	....	0.0240	0.0360	0.0144	0.0744
	1,900° F.						
1	0.044	0.70	....	....	0.0156	0.0144	0.0300
2	0.071	0.90	....	0.0132	0.0216	0.0024	0.0372
3	0.089	0.90	....	0.0144	0.0324	0.0144	0.0612
4	0.096	0.90	....	0.0180	0.0456	0.0096	0.0732
5	0.122	0.90	....	0.0204	0.0480	0.0192	0.0876
6	0.134	0.90	....	0.0360	0.0396	0.0252	0.1008
7	0.129	0.90	....	0.0312	0.0480	0.0144	0.0936
8	0.154	0.90	....	0.0492	0.0348	0.0216	0.1056
9	0.163	0.90	....	0.0564	0.0328	0.0180	0.0972
10	0.129	0.90	....	0.0048	0.0660	0.0324	0.1032
	Repeated Use at 1,750°.						
1st 10	0.1215	0.90	....	0.0240	0.0360	0.0144	0.0744
2d 10	0.083	0.90	....	0.0132	0.0468	0.0108	.0708
3d 10	0.060	0.70	....	....	0.0324	0.0132	0.0456
4th 10	0.047	0.70	....	....	0.0360	0.0168	0.0528

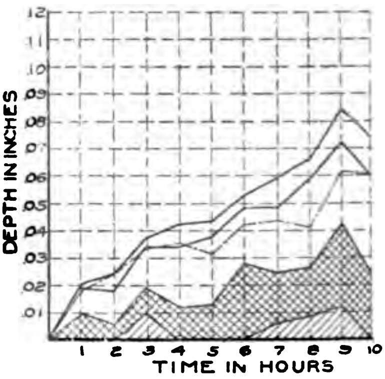
K-1600°



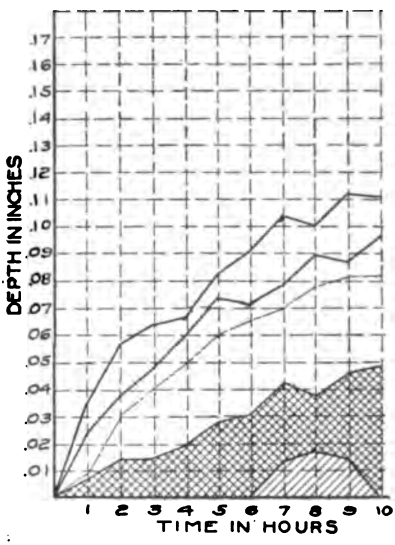
K-1750°



L-1750°



K-1900°



L-1900°

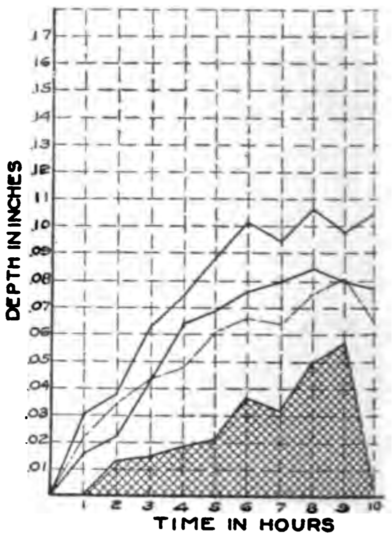
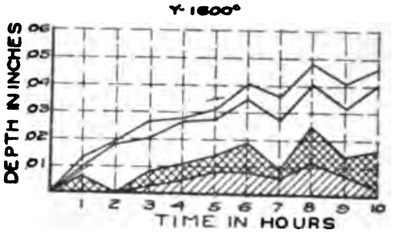
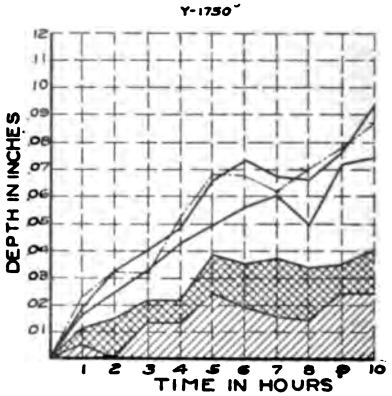
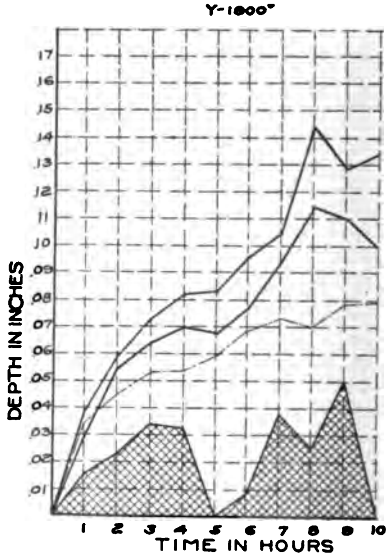


TABLE XIX.—*Material "M."*

Hours.	Increase in Weight.	Surface Carbon.	Depth of Carbon.				
			Over 0.90 Per Cent.	0.90 Per Cent.	0.90 to 0.50 Per Cent.	0.50 to 0.20 Per Cent.	Total.
1,600° F.							
	Grams	Per Cent.	Inches.	Inches.	Inches.	Inches.	Inches.
1	0.026	0.90	....	0.0060	0.0060	0.0012	0.0132
2	0.038	0.90	....	0.0108	0.0060	0.0012	0.0180
3	0.046	0.93	0.0060	0.0060	0.0060	0.0036	0.0216
4	0.054	0.92	0.0024	0.0084	0.0060	0.0072	0.0240
5	0.040	0.90	....	0.0060	0.0120	0.0072	0.0252
6	0.032	0.80	....	....	0.0180	0.0072	0.0252
7	0.050	0.90	....	0.0108	0.0096	0.0072	0.0276
8	0.073	1.05	0.0084	0.0084	0.0168	0.0084	0.0420
9	0.083	1.00	0.0108	0.0084	0.0240	0.0012	0.0444
10	0.089	1.00	0.0096	0.0156	0.0240	0.0084	0.0576
1,750° F.							
1	0.037	0.90	....	0.0084	0.0072	0.0060	0.0216
2	0.067	0.95	0.0072	0.0096	0.0096	0.0048	0.0312
3	0.088	1.10	0.0144	0.0108	0.0144	0.0072	0.0468
4	0.106	1.05	0.0180	0.0084	0.0228	0.0072	0.0564
5	0.112	1.10	0.0144	0.0168	0.0120	0.0108	0.0540
6	0.122	1.10	0.0204	0.0120	0.0204	0.0012	0.0540
7	0.121	1.10	0.0204	0.0132	0.0240	0.0084	0.0660
8	0.125	1.15	0.0144	0.0168	0.0228	0.0084	0.0624
9	0.124	1.05	0.0096	0.0204	0.0216	0.0072	0.0588
10	0.174	1.20	0.0300	0.0276	0.0216	0.0132	0.0924
1,900° F.							
1	0.061	0.90	....	0.0108	0.0168	0.0084	0.0360
2	0.094	0.95	0.0144	0.0108	0.0180	0.0132	0.0564
3	0.119	0.92	0.0084	0.0276	0.0264	0.0132	0.0756
4	0.142	1.00	0.0108	0.0312	0.0252	0.0180	0.0852
5	0.161	1.15	0.0180	0.0528	0.0168	0.0204	0.1080
6	0.178	1.10	0.0192	0.0468	0.0348	0.0132	0.1140
7	0.181	0.95	0.0228	0.0276	0.0528	0.0120	0.1152
8	0.204	0.95	0.0036	0.0468	0.0396	0.0192	0.1092
9	0.206	1.15	0.0156	0.0480	0.0408	0.0108	0.1152
10	0.223	1.00	0.0264	0.0420	0.0108	0.0192	0.1284
Repeated Use at 1,750°.							
1st 10	0.174	1.20	0.0300	0.0276	0.0216	0.0132	0.0924
2d 10	0.092	0.93	0.0036	0.0216	0.0360	0.0192	0.0804
3d 10	0.104	0.90	....	0.0216	0.0324	0.0072	0.0612
4th 10	0.092	0.90	....	0.0228	0.0432	0.0216	0.0876


Y-1600°
DEPTH IN INCHES
.06
.05
.04
.03
.02
.01
1
2
3
4
5
6
7
8
9
10
TIME IN HOURS


Y-1750°
DEPTH IN INCHES
12
11
10
.09
.08
.07
.06
.05
.04
.03
.02
.01
1
2
3
4
5
6
7
8
9
10
TIME IN HOURS


Y-1800°
DEPTH IN INCHES
17
16
15
14
13
12
11
10
.09
.08
.07
.06
.05
.04
.03
.02
.01
1
2
3
4
5
6
7
8
9
10
TIME IN HOURS

SUBJECT TO REVISION.

[TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS.]

The Constitution and Melting-Points of a Series of Copper-Slags.

BY CHARLES H. FULTON, CLEVELAND, OHIO.

(Cleveland Meeting, October, 1912.)

I. INTRODUCTION.

THERE are comparatively few accurate data on the melting- or the freezing-point temperature of metallurgical slags, or on related physical phenomena, such as fluidity near the melting-point, specific heat, latent heat, etc. With the exception of the work of H. O. Hofman,[1] and some older work by R. Åkerman, P. Gredt, and H. M. Howe, no information is available that is of direct interest to the metallurgist.

Hofman's valuable work is the only one that refers to slags produced in the metallurgy of copper and lead. The classic work of J. H. L. Vogt,[2] who views slags from the standpoint of physical mixtures (solutions in which the entities are minerals), has opened up a new field in the study of slags, and has placed the investigation of such properties as melting-temperatures, etc., on a rational basis. The recent work of A. L. Day, E. T. Allen, and W. P. White, of the Carnegie Institution, at Washington, and of European investigators[3] on certain systems of silicate minerals confirms the value of the investigation of silicate series from this standpoint, and it is my aim to apply this method to metallurgical slags, in order to gain exact data of value in metallurgy. The work recorded in the present paper is the result of an investigation of reverberatory-furnace slags made for a Western smelting company to determine exactly the influence of change in composition, within comparatively narrow limits, on the melting-point.

For a general discussion concerning the mineralogical character of slags and the principles on which this investigation is based, reference should be made to other work,[4] for the subject is too large to be stated in this paper, except very briefly in the following outline:

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[1] *Trans.*, xxix., 682 (1899).

[2] *Die Silikatschmelzlösungen; I. Ueber die Mineralbildung in Silikatschmelzlösungen; II. Ueber die Schmelzpunkt-Erniedrigung der Silikatschmelzlösungen* (Christiania, 1903–04).

[3] Doelter, *Handbuch der Mineralchemie* (1911

[4] C. H. Fulton, *Principles of Metallurgy.* Chapter viii., on Slags, p. 245 (1910).

Copper-slags, broadly considered, may have this range in composition: silica, from about 30 to 50; lime (including some magnesia), from about 2 to 35; ferrous oxide, from about 12 to 65; alumina, up to 12 per cent. Other bases, such as magnesia and zinc oxide, may be present in considerable amount. If so, the subject requires separate discussion and it is again referred to. When present in small amount, both magnesia and zinc oxide may be considered to replace iron oxide in some of the minerals formed. With the above range in composition the silicate degree of the slag varies between an ortho-silicate and a meta-silicate, and for practically all slags lies somewhere between the two. The slags when molten will consist largely of solutions of two minerals, one an ortho-silicate and the other a meta-silicate. What these minerals are depends upon the base composition of the slag, very largely upon the relation of lime to iron oxide to alumina. In this manner series are formed in which the two minerals are respectively the entities or end-members, and all compositions between are mixtures, in various proportion, of the two minerals. The addition of one mineral to the other will cause a change in the freezing-point according to well-known laws, and for the range in composition from one mineral to the other there is established a freezing-point or a melting-point curve which is rational and will fall into one of the well-known types.

In Table I. are given the mineral series possible for copper- and lead-slags, according to silicate degree and base composition.

TABLE I.—*Mineral Series of Copper- and Lead-Slags.*

Silicate Degree.	Mineral Series.	Base Composition Within Limits Specified.
Ortho-silicate	1. Olivine—magnetite (fayalite)	Very high in iro.
Ortho-silicate	2. Olivine—melilite	High lime, comparatively low iron. High alumina.
Between ortho- and meta-silicate.	3. Melilite—augite	Same as above.
	4. Augite—olivine (iron-lime olivine)	Somewhat less lime, more iron ; high alumina.
	5. Augite—olivine (fayalite)	Comparatively high iron, still less lime, lower alumina.
	6. Rhombic pyroxene-olivine (hypersthene)—(fayalite	Very little lime, high iron, low alumina.
Meta-silicate	7. Rhombic pyroxene (hypersthene)—augite	Low lime, comparatively high iron; some alumina.

The slags ordinarily made in copper-smelting will fall chiefly into series 3, 4, 5, and 6, dependent on the base composition. Alumina in the presence of considerable lime will go to form augite or melilite. If the composition of the slag mixture is such that neither of these minerals can form, alumina is very apt to form spinel, which crystallizes out of the molten slag at a high temperature, causing difficulties. The formation of spinel is greatly aided by the presence of magnesia or zinc, or both, even in small quantities. Ordinarily magnesia passes to the olivine or hypersthene mineral, zinc to the olivine mineral, particularly to the high-iron olivine or fayalite. It is thus apparent that if a slag is to carry considerable alumina it must also contain considerable lime in order to form augite or melilite, and if it is to carry considerable zinc, much iron and but little lime must be present to give opportunity to form fayalite. High alumina and zinc are incompatible. High alumina and magnesia are incompatible.

It is to be noted that the minerals named are used to designate group minerals. Thus olivine may vary widely in composition, including such minerals as fayalite, true olivine, an iron-lime olivine, etc. The same is true of the other minerals named. For the purpose of a determination of melting-points and related properties it is proposed to investigate the series outlined in Table II.

TABLE II.—*Base Composition of Slags.*

Silicate Degree.	Base Composition.	Mineral Series.
Silica varying between ortho- and meta-silicate....	1. $\frac{CaO}{FeO} = \frac{70}{30}$ 2. $\frac{CaO}{FeO} = \frac{60}{40}$ Alumina up to 18 per cent.	Melilite—augite.
Silica varying between ortho- and meta-silicate.....	3. $\frac{CaO}{FeO} = \frac{50}{50}$ 4. $\frac{CaO}{FeO} = \frac{40}{60}$ Alumina up to 8 per cent.	Olivine—augite.
Silica varying between ortho- and meta-silicate...	5. $\frac{CaO}{FeO} = \frac{30}{70}$ 6. $\frac{CaO}{FeO} = \frac{5}{95}$ Alumina below 6 per cent.	Olivine —augite. —hypersthene.

If the melting-points of a considerable number of slags in these six series be determined, the field of copper-slags will be practically covered, for additional curves of interest can then be plotted from the data obtained by taking the slags of the same silica-content from the six series, showing the effect of varying ferrous oxide and lime with constant silica. The effect of varying alumina will have to be separately determined.

In the work described in the succeeding sections of the present paper one olivine-augite series and two melilite-augite series were investigated, in which in each series the iron-alumina ratio remained constant, while the lime varied with the silicate degree. The reason for this is that the ordinary reverberatory-slags fall into such a series, and that the problem presented was a practical one, viz.: with a certain ore-mixture essentially constant in composition with regard to silica, iron, and alumina, what is the effect on the melting-point of the slag formed of the addition of increasing quantities of lime (the flux)? Also, how low may the lime be, and how high can the silicate degree be carried, and still have practical slags?

II. CONSTITUTION OF CERTAIN REVERBERATORY-SLAGS.

In investigating the problem it was thought desirable to examine the slags from the reverberatory furnaces from the chemical and mineralogical standpoints. Table III. gives in a concise way the compositions of these slags, which were produced by smelting calcines obtained by roasting concentrates mixed with crushed limestone, with a certain amount of converter-slag, from both basic- and acid-lined converters. The analyses represent very careful work with accurate and exact methods, not the methods used in ordinary rapid smelter-work. The regular slags tapped from the furnace are designated as "clean slags." "Floaters" are masses of loosened furnace-bottoms. "Blanket slag" is a slag which forms on the surface of the charge and which seems to consist of partly-fused material. The following additional petrographic notes elaborate the identification of the minerals. (I am indebted to Dr. Charles P. Berkey for the identification of the minerals in the slags).

TABLE III.—*Composition of Reverberatory-Furnace Slags.*

No.	Type.	Composition.										Petrographic Notes.
		SiO_2.	Fe_2O_3.	Fe.	Al_2O_3.	CaO.	MgO.	Cu.	n.	K_2O Na_2O.	Total.[a]	
		P. C.	Per Cent.	P. C.	P. C.	P. C.	P. C.	P.C.	P.C.	P. C.	P. C.	
2	Blanket slag........	38.68	42.36 FeO = 38.10	29.65	8.48	9.92	0.79	......	......	1.00 ?	96.97	Ground-mass essentially a brownish glass. Small feather-like crystals of a pyroxene (augite). Small pieces of unaltered quartz. Numerous matte globules. Flow structure.
3	Blanket slag......	52.08	31.08 FeO = 27.96	21.76	9.00	5.90	0.23	......	......	1.00 ?	96.17	Main mass essentially a brown glass. Flow structure. Considerable magnetite. Some small feather-like crystals of augite. Unaltered quartz. Numerous matte globules.
9	Floater slag.......	84.62	9.48 FeO = 8.53	6.64	2.13	0.90	Tr.	......	......	1.00 ?	97.18	Ground-mass a glass. Much original quartz. Many rod-like colorless crystals, perhaps olivine? Some matte globules.
12	Clean slag.........	38.50	42.20 Fe_3O_4 = 40.80 FeO = 37.98	29.54	7.62	12.62	1.01	0.21	......	1.74 0.50	99.97	Ground-mass a glass. Large number of feather-like crystals of a pyroxene, probably hedenbergite. Some colorless rod-like crystals, as above. Considerable magnetite. Matte globules.
13	Clean slag.........	42.74	35.94 FeO = 32.33	25.16	7.76	13.32	0.94	......	0.60	1.00 ?	98.09	Mostly an aggregate of green fern-like crystals of a pyroxene (augite), constituting about 75 per cent. of the slide. Considerable magnetite. Numerous very small matte globules.
14	Clean slag.........	40.12	40.47 FeO = 36.40	28.33	7.21	13.22	0.98	......	......	1.00 ?	98.93	Same description as for No. 13.
15	Clean slag	37.24	47.68 FeO = 42.92	33.38	6.38	10.32	0.96	......	......	1.00 ?	98.82	Mostly an aggregate of light green radiating, interpenetrating crystals of probably hedenbergite and a darker light brown augite. Much magnetite, some matte globules.
16	Crystalline slag...	49.96	29.67 FeO = 26.69	20.77	8.90	12.12	1.30	......	......	1.00 ?	99.97	Consists largely of augite, well crystallized, the balance being magnetite and a little glass.

[a] Exclusive of Zn, Cu, and S. Iron as FeO.

Slag No. 8.—A dark slag with many roundish light-colored grains. One mineral is present in considerable amount. It is colorless and occurs in round, corroded grains that have neither crystal-boundaries nor internal structural features. The grains are fractured. The mineral has good relief and low double refraction. It is uniaxial and positive. It is evidently the original undecomposed quartz that has not been fluxed. The ground-mass is glassy, shows strong flowage, and is brown to black in color. Those portions showing the strongest flowage are brown in color, and portions immediately surrounding the crystals grains are generally brown. The rest of the field is very dark and carries an abundance of black metallic particles, very likely magnetite. The lighter color about the quartz grains seems to show the solution of the quartz into the glass or slag. In a few places large metallic grains occur, probably matte globules. Certain areas show no flowage at all, but have a very minute micro-granular structure that is the beginning of crystallization. In limited areas this gives a mottled effect similar to that produced by the devitrification of natural glasses. These areas have a yellowish or greenish-yellow color, and a refraction and double refraction of higher order than other areas. They are probably the beginnings of pyroxenes.

Slag No. 9.—An abundance of crystalline minerals is present. The minerals grow in rods, or plates, which are in bundles or aggregates, and constitute about one-third of the area. It is of moderate relief, entirely colorless, strong crystallizing tendency, of low double refraction, parallel extinction in all cases, uniaxial and positive. The latter characteristics are not very readily made out, as the interference figure is indistinct. The elongation is parallel to the "a" elasticity axis. The slag has a striking structural appearance. The rod-like structure is strongly developed, with numbers lying parallel and with inclusions of the dark glassy matrix in and between them. They are associated with other groups lying at various angles and larger aggregates of less definite boundaries. The mineral is not identified. The rest of the slide consists of comparatively large areas of original quartz, unfluxed, and a greenish-brown glass. Minute particles like mineral dust are also present.

Slag No. 12.—The main body of this slag is divided between an opaque glass and a glassy green mineral. In very thin section the color is green to brown, and the mineral is pleochroic. The extinction angle when measured from the central shaft of a fern-like crystallite is often high, up to 30°. Refraction and double refraction are strong. Interference colors are of the second and third orders in thin sections. There is little doubt that the mineral is a pyroxene, probably near hedenbergite. In some of the better formed crystals there are lines of opaque inclusions of a yellow-gray and a steel-gray metallic substance, probably magnetite and sulphides (matte). Magnetite also occurs in little patches with rounded irregular outline, forming a considerable proportion of the slag (perhaps from 5 to 10 per cent.). There are some oblong crystals of a nearly colorless mineral with cleavage parallel to the elongation and extinction parallel to the cleavage. This corresponds to a member of the olivine group, although the double refraction seems to be less than in natural olivine. The slide tested with acid shows gelatinization, pointing to the presence of olivine. The rod-like crystals are of the same kind as in slag No. 9.

Slag No. 14.—In this slag there is enough crystalline structure to show that some mineral constituent is very generally developed in the slag. The structure shows itself in branching and feather-like aggregates, and in areas where the components have strong individual elongation. The mineral is light green in color. It shows brownish pleochroism and an abundance of dark inclusions which accentuate the structure. It is biaxial with oblique extinction and high double refraction and strong relief. The extinction angle varies from zero to more than 40°. These are the characteristics of the colored monoclinic pyroxenes. The color undoubtedly indicates that iron is a prominent constituent. The mineral corresponds more nearly to common augite than to any other variety. No other mineral is developed in this slide except magnetite, which is plentifully distributed about the branching forms of the augite. This last constitutes about 75 per cent. of the slag.

Slag No. 15.—This slag shows crystalline development in long needles of radiating, branching and interpenetrating structure, and the matrix between them shows also the characteristics of the incipient crystallization. The crystals and aggregates are light brown to light green in color. The only other individualized constituent occurs as minute, metallic black grains, some of which are apparently octahedral in form and very abundant throughout the section. This mineral is magnetite. The light-colored crystals appear to be of two distinct varieties. The one most strikingly developed is a very light green, and is in the form of long needles with feather-like branches or cross-bars. It is always surrounded and intergrown with an excessive amount of black granular material, probably magnetite, which is more abundant in its immediate vicinity than in other parts of the section. This mineral is a pyroxene, and so far as tests can be made agrees well with hedenbergite, a non-aluminous variety. The other constituent is light brown in color and not so clear. It constitutes a matrix of interpenetrating rods and feather-like aggregates. Throughout this aggregate the magnetite grains are scattered plentifully. Optical tests indicate that this mineral is also a pyroxene, and that it belongs to the colored monoclinic varieties included under augite. There is apparently no great difference between these two constituents in classification, but they have strikingly different development, which points to difference in chemical composition. The last mineral constitutes fully one-half of the slag. By reflected light very small matte particles can be discerned scattered throughout the mass. There are some rod-like crystals similar to those found in slag No. 9.

Slag No. 16.—This specimen is crystalline and made up of two constituents, one of which is pyroxene and the other magnetite. Very small areas are isotropic and are probably included glass. The prominent mineral occurs in long columnar growths, which in thin sections appear either as long crystals with irregular margins and very prominent longitudinal structure, due to multitudes of inclusions, or as a geometrically branching or graphic area which represents the same type of crystal cut cross-wise. The grains cut longitudinally have a green to yellowish green pleochroism, those cut cross-wise vary from green to yellowish brown. The mineral has a high extinction angle and double refraction, and the optical characteristics are those of ferriferous augite (or basaltic augite). This mineral constitutes about 75 per cent. of the slag. The rest is magnetite and undifferentiated residual glass, both of which occur as inclusions, but more especially as a sort of matrix which surrounds every crystal and fills the grooves or longitudinal re-entrant angles along their fluted sides.

On a critical analysis of the results set forth it is apparent that the most prominent constituent of the regular slags is a pyroxene, in some instances hedenbergite, in most cases augite. In one slag both are present. Olivine is also probably present and magnetite is a common constituent. These slags belong to the binary series, augite-olivine, in which, according to J. H. L. Vogt,[5] a eutectic is present, or they may be considered to belong to a ternary series, augite-olivine-magnetite. In the first-named series the eutectic composition, or the mixture of the two minerals that has the lowest melting-point, occurs when the silicate degree of the mixture is about 1.65. In the freezing of a slag, a member of this series, either augite or olivine will crystallize out first as the "excess substance," *i.e.*, that which is present above that required for the eutectic ratio, dependent upon whether the silicate degree of the slag be greater or less than 1.65. While the eutectic in metallic alloys consists of the conglomerate of the two entities which form, in a silicate, owing to viscosity, the eutectic usually congeals to a glass (the residual glass of the above description) and one member of the series, in this instance the olivine, may be largely in the glass and undistinguishable as a distinct mineral. The presence of an abundance of augite and practically no olivine shows these slags to belong on the augite side of the eutectic, and of a silicate degree greater than 1.65. This is not apparent from the chemical analysis, for calculations made in the ordinary manner, with all the iron present as ferrous oxide, give a silicate degree considerably lower; *e.g.*, the silicate degree of slag 15 is 1.22, which calls for olivine crystallization first.* This apparent discrepancy is readily explained when the presence of magnetite is considered. A very considerable portion of the iron shown by the analyses is in the form of magnetite. This has the effect of increasing the silicate degree of the slag very decidedly above that called for by the analysis. The source of this magnetite in the slags is not certain. It probably comes from the converter-slag which is added in the furnaces, but part of it may be in the calcines of the charge, or be formed during smelting. The iron in the magnetite cannot be considered as flux for silica, and its presence is detrimental in so far as it crystallizes out of the slag at comparatively high temperatures, passing in part to the underlying matte, and when present in sufficient quantity making a mushy slag, which may entangle considerable matte and thus cause high-copper slags. The value of converter-slag as a flux seems questionable in many instances on account of the presence of this magnetite. Magnetite is present in many slags high in iron.

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[5] *Ueber die Schmelzpunkt-Erniedrigung der Silikatschmelzlösungen*, 114. Doelter, *Handbuch der Mineralchemie*, vol. i., p. 943 (1911).

* If 25 per cent. of the iron present in this slag is in the form of magnetite, the silicate degree becomes 1.45.

It is customary to consider the iron in slags to be present in the ferrous condition entirely, and combined with silica. That this is not the case can readily be seen from the results of the analyses in Table III., most of which do not add to 100 per cent. when the iron is calculated as ferrous oxide.* This, of course, is confirmed by the microscopic analysis. It has been noted frequently by different observers that many slags containing considerable iron are magnetic. It is also worthy of note that slags containing augite very likely contain ferric iron as well as ferrous iron, since both forms of the oxide enter into the constitution of the augite molecule. That this is the case with the reverberatory-slags under discussion is indicated by the brownish color of some of the augite found. The amount of magnetite found in the slags was sometimes very appreciable, more than 10 per cent. of the weight of the slag, thus removing a large portion of the iron from the influence of silica. This description gives some idea of the complex condition of the iron in some slags, and the effect it has on the nature of the slag.

III. Melting-Points of Slags.

In the discussion of certain thermal properties of slags such terms as "formation-temperature," melting-point, freezing-point, viscosity, superheating, and fluidity are frequently used, sometimes in a confusing manner, so that it seems desirable to view some of these properties in the light of recent investigation. From the metallurgical standpoint the most important physical property of a slag is the temperature at which it will readily flow from the furnace and permit the separation of metal and matte from it by gravity. This property of fluidity is closely related to the melting-point and to the viscosity (resistance to flow) in a certain temperature-range above the melting-point. The melting-temperature or point, defined as that temperature at which the solid slag passes to the liquid condition, is perhaps the property best suited for the purpose of investigation, as will be evident from what follows. The melting-point of a silicate, however, presents different aspects than the melting-point of metals or substances which are excellent conductors of heat and of low viscosity. When a mass of metal to which heat is being supplied from an exterior source at a sufficient rate begins to melt on the outside layers, the conduction of heat to the interior of the mass is so rapid that all of it practically reaches the melting-point at the same time. This is not true of silicates. In this case the melting of the mass takes an appreciable time and the time-temperature curve during the melting-stage is therefore an oblique one, and not the sharp, nearly horizontal jog that shows in the freezing-curve or melting-curve of metals. This time-lag in the melting or freezing process is most pronounced the more viscous the substance is at or near the melting-point and is very prominent in such substances as orthoclase and albite, or other silicates containing considerable alumina, and in some borates, but is not very pronounced in the iron, lime, or manganese silicates. Many silicates also possess the property of very appreciably undercooling below the true freezing-point before solidification or crystallization sets in and the latent heat of fusion is given out. Therefore in taking a freezing-point this almost invariably comes too low, sometimes by 100° or more. Moreover, the amount of lowering is not constant and may vary with each determination of the freezing-point. I spent several months fruitlessly in endeavoring to work out the slag series which follow by the freezing-point method. The data in Table IV. and in Fig. 1 illustrate these facts.

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* Slag No. 15 contains 33.38 per cent. of iron. If 25 per cent. of this be calculated as Fe_3O_4, and 75 per cent. as FeO, the total of the slag-constituents will be increased by 0.79 per cent. only.

TABLE IV.—*Melting- and Freezing-Curves on Slag No.* 16.

Melting-Curves.						Cooling-Curves.			
Mic. Volts.	Dif.	Mic. Volts.	Dif.	Mic. Volts.	Dif.	Mic. Volts.	Dif.	Mic. Volts.	Dif.
7,000	90	6,730	140	7,730	170	10,910	210	9,600	200
7,090	90	6,870	140	7,900	160	10,700	240	9,400	300
7,180	90	7,010	140	8,060	160	10,460	240	9,100	200
7,270	90	7,150	130	8,220	150	10,220	220	8,900	200
7,360	80	7,280	130	8,370	120	10,000	230	8,700	150
7,440	80	7,410	130	8,490	110	9,770	210	8,550	150
7,520	80	7,540	130	8,600	95	9,560	210	8,400	200
7,600	80	7,670	120	8,695	80	9,350	190	8,200	200
7,680	90	7,790	115	8,775	75	9,160	180	8,000	150
7,770	80	7,905	115	8,850	60	8,980	180	7,850	......
7,850	80	8,020	110	8,910	50	8,800	......	......	......
7,930	90	8,130	100	8,960	60				
8,020	80	8.230	100	9,020	55				
8,100	80	8,330	100	9,075	55				
8,180	90	8,430	80	9,130	70				
8,270	90	8,510	90	9,200	70				
8,360	70	8,600	80	9,270	80				
8,430	70	8,680	70	9,350	100				
8,500	70	8,750	60	9,450	110				
8,570	70	8,810	50	9,560	140				
8,640	60	8,860	50	9,700	230				
8,700	60	8,910	50	9,930	120				
8,760	50	8,960	50	10,050	110				
8,810	40	9,010	50	10,160	95				
8,850	50	9,060	40	10,255	85				
8,900	40	9,100	50	10,340	75				
8,940	40	9,150	55	10,415	65				
8,980	50	9,205	55	10,480	......				
9,030	40	9,260	70	......	......				
9,070	40	9,330	90						
9,110	40	9,420	85	c. j.—28° C.					
9,150	50	9,505	70	......	......				
9,200	60	9,575	95	......	......				
9,260	60	9,670	140	......	......				
9,320	80	9,810	170	......	......				
9,400	80	9,980	135	......	......				
9,480	90	10,115	95	.....	......				
9,570	90	10,210	90	......	......				
9,660	90	10,300	80	......	......				
9,750	160	10,380	70	......	......				
9,910	130	10,450	......	......	......				
10,040	......	......	......	......	......				
c. j. – 22° C.		c. j.—28° C.							

Melting-points:

No. I. = 1,036° C.
No. II. = 1,036° C.
No. III. = 1,037° C.

NOTE.—Readings at minute intervals

On slag No. 16, which is very nearly a bisilicate, only the melting-curve gives results. This slag is so viscous near its freezing-point that practically no inflection is to be noted in the cooling-curve. Somewhat better results are obtained with the cooling-curve if the rate of cooling is less than shown in the table, but they are far from conclusive. It may therefore be stated that the investigation of the freezing-points of slags is much less satisfactory than that of the melting-points, due to the great undercooling which takes place.

While undercooling[6] is a prominent phenomena in silicates, overheating—viz., heating beyond the temperature of the true melting-point without liquefaction—is also possible, according to Doelter,[7] for many silicates, and, according to Day and Allen,[8] for certain very viscous materials, such as albite, orthoclase, or quartz. According to W. P. White,[9] it is negligible for many silicates, especially for those of the types under consideration.

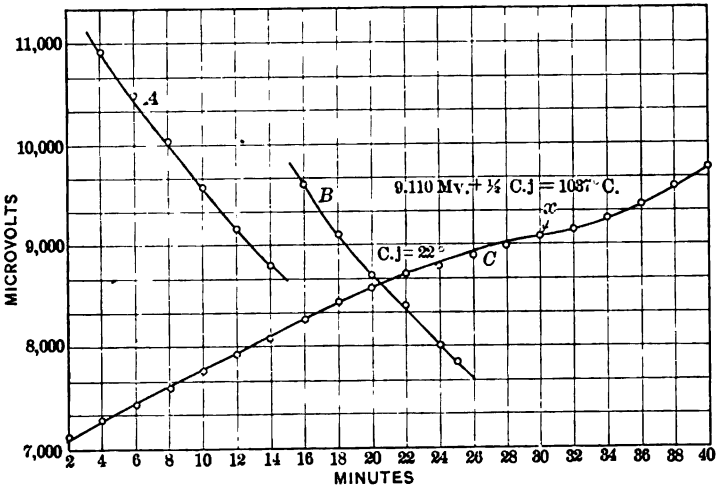

FIG. 1.—FREEZING- AND MELTING-CURVES OF SLAG NO. 16, ELECTRIC FURNACE.

The experimental evidence of this fact rests in being able to get the same melting-point repeatedly with different rates of heating. If overheating can take place, the melting-point comes higher with the increased heating-rate. In the work under consideration rather widely different heating-rates gave but very little change in the melting-point.

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[6] W. P. White, Melting-Point Determination, *American Journal of Science*, Fourth Series, vol. xxviii., No. 167, p. 455 (Nov., 1909). C. Doelter, *Handbuch der Mineralchemie*, vol. i., p. 677 (1911).

[7] *Handbuch der Mineralchemie*, vol. i., p. 631 (1911).

[8] The Isomorphism and Thermal Properties of the Feldspars, *American Journal of Science*, Fourth Series, vol. xix., No. 110, p. 119 (Feb., 1905).

[9] *Ibid.*

The term "formation-temperature" of a slag is usually taken to mean the temperature at which the mixture of constituents that go to form the slag becomes fluid. When closely considered this is a vague conception, for this temperature is a variable, depending on a number of factors, viz., rapidity of heating, the possible formation of a eutectic, size of particles of the mixture. It is also possible that in the mechanical mixture one substance may go partly into solution in the others, below the melting-point of the mass as a whole. The formation-point, considered as the temperature at which chemical combination takes place, may lie below the melting-point, as has been shown by Day and Shepherd for calcium silicates.[10] From the metallurgical standpoint, and with coarse material, the formation-temperature, so-called, lies above the melting-point, but it lacks definiteness for the purpose of investigation.[11] The upper melting-point of the formed slag (viz., not that of the eutectic, if such be present, but that at which the whole mass passes to the fluid state) is a definite and fixed point with those silicates in which overheating is not a phenomenon, and for that reason was adopted as the basis of this investigation. The viscosity of the slags under consideration increases with the silica- and alumina-contents and varies much for the different slags. For this reason it is necessary, aside from the melting-point, to determine what temperature is required to cause the slags to be sufficiently limpid to flow, *i.e.*, what amount of superheating is required above the melting-point. This degree of fluidity was determined by stirring with a heavy platinum rod and noting the temperature reading at the time of stirring.

The method of taking melting-points used in this investigation is based on the researches of Walter P. White,[12] to which reference should be made for a complete discussion of the subject. Briefly stated, the essential data are as follow: (1) A furnace is necessary in which the heating-rate is under perfect control. A properly-designed electric furnace is best for the purpose.

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[10] *American Journal of Science*, Fourth Series, vol. xxii., No. 130, p. 268 (Oct., 1906).

[11] *Principles of Metallurgy*, p. 272 (1910). *Handbuch der Mineralchemie*, vol. i., pp. 646, 954 (1911). H. O. Hofman, *Mineral Industry*, vol. xvii., p. 597 (1908).

[12] Melting Point Determination; Melting Point Methods at High Temperatures, *American Journal of Science*, Fourth Series, vol. xxviii., No. 167, pp. 453 to 489 (Nov., 1909).

(2) Accurately standardized thermo-couples are required. The hot-junction should be properly centered in the mass, the melting-point of which is to be determined. (3) It is essential to have means of accurately measuring small differences of potential generated in the thermo-couples. This is best done by means of a potentiometer and a delicate reflecting galvanometer.

The hot-junction of the thermo-couple is centered in the molten substance contained in a platinum crucible, being placed two-thirds down from the top of the molten mass. The contents of the crucible are then allowed to solidify within the furnace, care being taken to have the crucible so placed that it will heat uniformly from the walls of the furnace when the crucible is reheated. The particular position in the furnace is best determined by experiment. The thermo-couple always indicates the temperature of a small mass of material immediately surrounding the hot-junction of the couple.

Two methods of taking heating-curves are available. The current of the furnace is so adjusted and varied that the amount of heat supplied to the mass to be melted is constant. If the current on the furnace is kept constant, which means essentially that the heating-rate of the furnace will be constant, then the amount of heat supplied to the melting mass will be variable, for the amount of heat which passes to the mass from the furnace-walls depends upon their difference in temperature. This difference determines the heating-rate. At the beginning of the heating there is a certain difference in temperature between the furnace-wall and the crucible contents. When melting commences heat is absorbed by the crucible contents and its temperature-rise diminishes and falls behind that of the furnace-wall. Hence the temperature difference between furnace and crucible increases as the melting proceeds, and the rate of heat-supply is rapidly increased towards the end of the melting, with the result that this portion is greatly hurried, the melting-curve becoming oblique, and the rise or break in the curve after the melting of the material around the junction of the couple is not as sharp as it would be if the rate of heat-supply had been less. In fact, with too high a heating-rate the break may be much obscured. For examples of heating-curves with a variable heat-supply, in the manner just discussed, but with a comparatively low rate, consult Fig. 2, that of copper, and Fig. 3, that of salt. During the melting of the mass immediately surrounding the hot-junction of the thermo-couple the heating-rate is practically *nil* in substances that are good conductors and not viscous, but in substances like silicates the rate decreases to a minimum, to be followed by a rise, as in the melting-curve of slag No. 3 in Fig. 3. The more rapid rise suc-

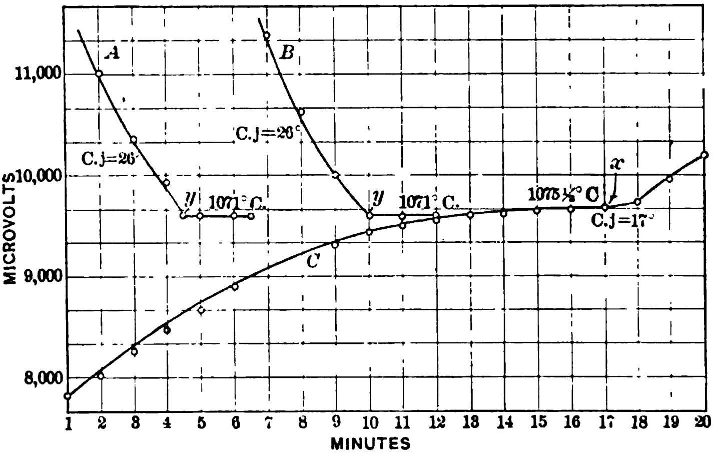

FIG. 2.—FREEZING- AND MELTING-CURVES OF COPPER.
Freezing-Points in Gas-Furnace; Melting-Point in Electric Furnace; Copper at 1,034° C.

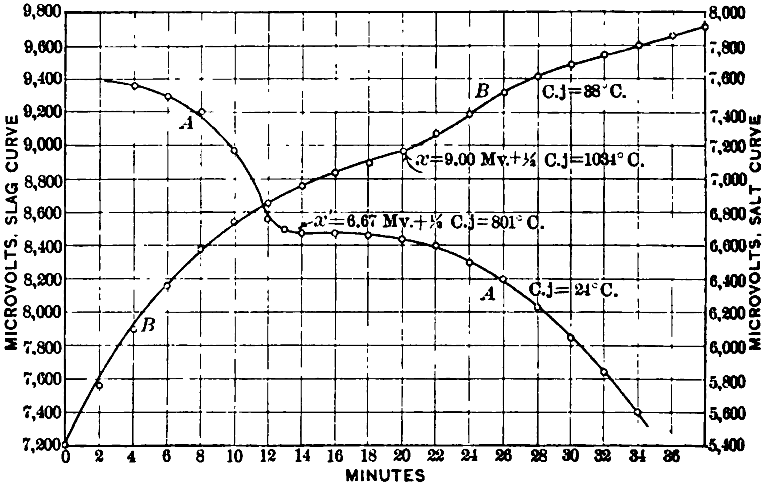

FIG. 3.—*A* = MELTING-CURVE OF SALT. *B* = MELTING CURVE OF SLAG NO. 3, SERIES III.

ceeding the initial rise is probably due to convection-currents within the molten mass. The method of heating by variable heat-supply distorts the heating-curve as shown, and while by this method the upper melting-point can be determined within a few degrees if the heating-rate be properly adjusted, other melting-points, such as that of the eutectic, are obscured. Greater accuracy is obtained if the furnace is operated to maintain a constant difference of temperature between the furnace-wall and the melting mass at all stages of the melting. This can be done by means of a second or control couple placed in the furnace-chamber next the crucible at about the same level. The difference in reading between this couple and that in the melting mass is kept constant, or nearly so, by controlling the current supplied to the furnace by means of a rheostat. The resulting curve is much less oblique than that taken by the first method and more sharply defines the rise at the end of the melting, thus placing the melting-point more accurately, and also serves to give an accurate melting-curve, on which eutectic or residual meltings can be determined.

If the heating-rate be so adjusted, with a constant furnace-temperature, so as not to be too high during the latter part of the melting interval, a curve results on which the break due to the melting of the mass surrounding the junction of the couple is sharp and pronounced and the melting-point determined within a degree or two. As the upper melting-points only were sought for in this work this method was adopted as sufficiently accurate. The same point may be obtained repeatedly, except when the heating-rate is too high. Then it will come a few degrees above the true point. Table III. gives three heating-curves on slag No. 16, the last two of which are taken with somewhat too high a rate; still, the displacement in the melting-point is small. Curve No. 1 of this slag is plotted in Fig. 1. In Fig. 3, *B* shows the heating-curve of slag No. 3 in series III. Heating-curves of slags in series I. and II. are shown in Figs. 4 and 5, respectively. The records of the heating show the melting-point better than the plotted curves. The melting-point is taken as lying between the reading recording the lowest rate and that recording the first increase thereafter. Very often a sharp deflection of the beam of light on the galvanometer scale is observed when the mass surrounding the junction melts. Thereafter the rise in rate may be either comparatively slow at first and then increase suddenly, or it may increase rapidly at once, dependent on the viscosity of the molten slag.

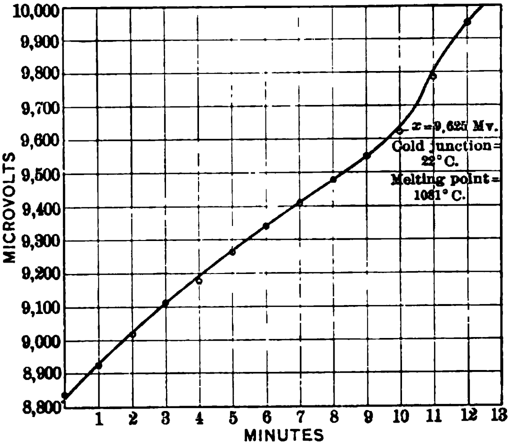

Fig. 4.—Melting-Curve of Slag No. 4, Series I.

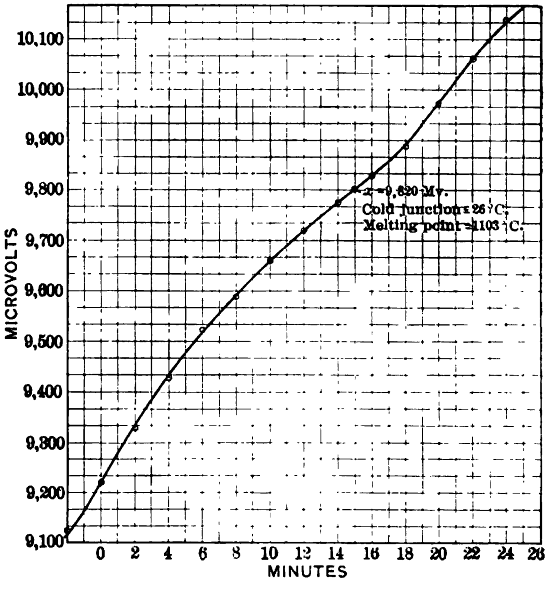

Fig. 5.—Melting-Curve of Slag No. 8, Series II.

If viscous, the rise is comparatively slow; if limpid, the rise is rapid. This leads to the conclusion that the rapid rise in rate is due to the equalization of temperature brought about by convection-currents, which are delayed in the viscous material until it becomes sufficiently superheated.

The Furnace.—The most suitable furnace for obtaining uniform heating-conditions is some form of electric-resistance furnace. In this work it was expected that temperatures above 1,400° C. would be required. Most furnaces on the market, with the exception of platinum-wound furnaces, are not available, as this temperature is beyond that for which they are constructed. This refers to furnaces in which the heating-rate is under such control as to permit their use for scientific investigation. A foreign carbon-resistance furnace was imported, but proved unsuitable in the work. After considerable experimental work the furnace shown in detail drawing in Fig. 6 and in perspective in Fig. 7 was built, and gave most excellent service. The resistor is kryptol. The cross-section and length of the resistor cylinder are designed for a 115-volt direct current. The current-consumption is from 10 to 30 amperes,

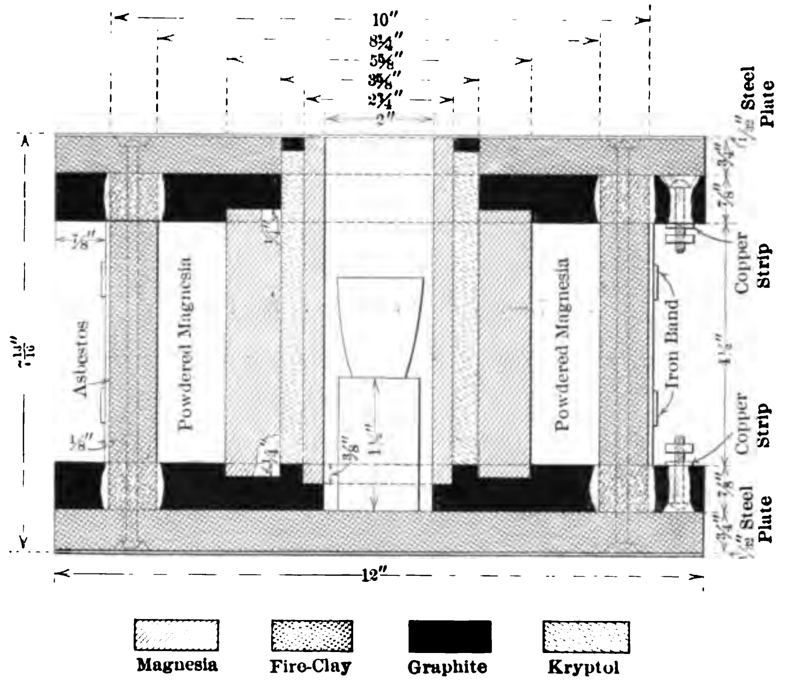

FIG. 6.—ELECTRIC-RESISTANCE FURNACE.

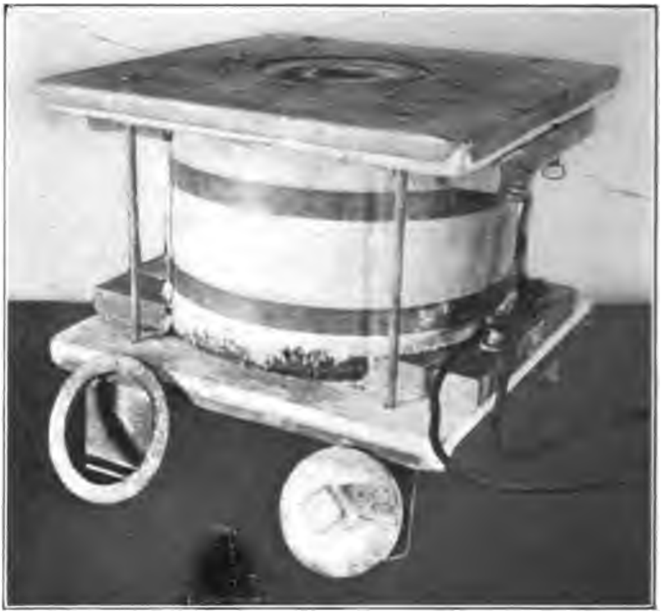

FIG. 7.—ELECTRIC FURNACE. KRYPTOL.

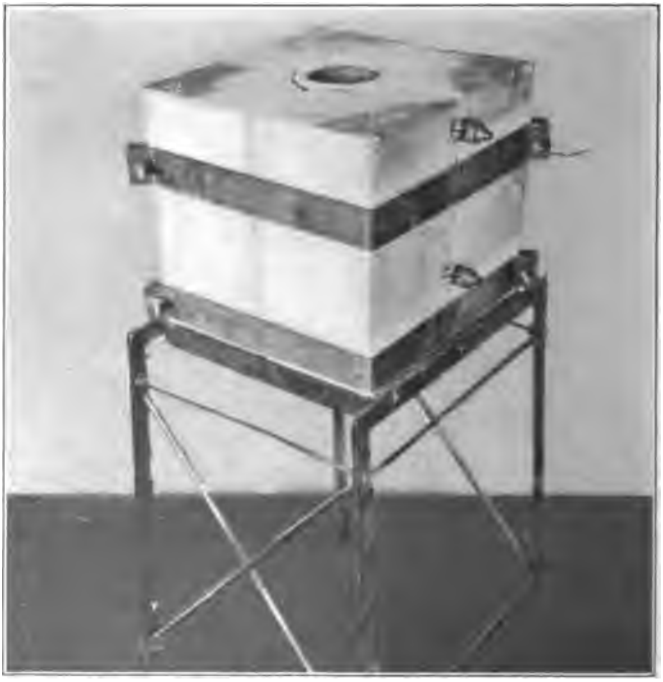

FIG. 8.—ELECTRIC FURNACE. PLATINUM WIRE.

dependent upon the temperature to be obtained. With the latter current the limit of the platinum-rhodium thermo-couple is reached in about 4 hr., if the furnace is protected against excessive radiation. The essential features[13] of the furnace are the following: (1) Two concentric cylindrical tubes, the inner one of which forms the heating-chamber, made of either magnesia (calcined magnesite furnished by the Harbison-Walker Co., Pittsburg) or alundum, and the cylindrical space between which, when filled with a resistor like kryptol, becomes the resistor cylinder. (2) Uniform and continuous contact of the resistor on its whole cross-section with the lower graphite electrode plate. (3) A resistor of uniform cross-section throughout its length. (4) The positive and firm insertion of the tubes in the electrode plates, thus maintaining the resistor chamber in perfect shape. (5) That any part can be readily and cheaply renewed. (6) Simplicity and compactness of construction, preventing racking and distortion during heating.

A practical difficulty experienced with most furnaces using a granular carbon resistor is the uneven heating of the furnace space, usually due to poor design of the shape of the resistor space. The furnace described gives very little trouble in this respect, and then only when, after considerable use with high currents, arcing takes place locally in the resistor, destroying the kryptol and fusing small portions of the tubes. An error in the design of many resistance-furnaces seems to be in the effort made to operate with too small an amount of current, making the cross-section of the resistor, either granular or wire, so small that local overheating, with consequent burning-out, readily takes place. This saving of a few watts of current is more than made up in expense for repairs and annoyance.

A furnace of the type used by Day and Allen[14] was also used for part of the work. It consists of a magnesia tube 6.5 in. high and 2.25 in. inside diameter, on the inside of which in a spiral groove are wound 25 coils of platinum wire, No. 18 B & S. gauge. This wire is 17 ft. long and 0.0403 in. in diameter and weighs 91.2 g. The furnace was designed for a maximum current of 45 amperes at a voltage of 115. For the work in question a current of from 10 to 20 amperes was used. The tube is inclosed in fire-clay blocks, as shown in Fig. 8. This particular model was designed to permit of pouring the contents of crucibles within the furnace directly into a calorimeter, after opening a slide door which closes the bottom of the heating-tube. By this method specific heats of slag were determined. This furnace is eminently satisfactory, but the cost, at the present price of platinum, is very high.

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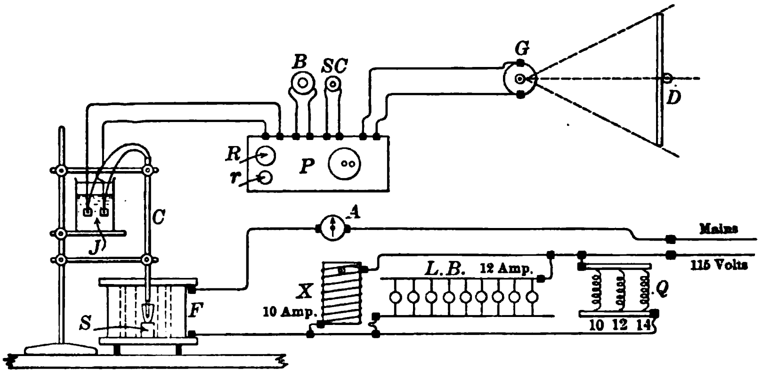
[13] Patent applied for by C. H. Fulton and W. A. Coursen.

[14] Temperature Measurements to 1600° C., *Physical Review*, vol. xix., No. 3, p. 177 (Sept., 1904). *American Journal of Science*, Fourth Series, vol. xxvi., No. 155, p. 411 (Nov., 1908); vol. xxviii., No. 167, p. 474 (Nov., 1909).



F, Furnace.
J, Cold-junction.
S, Crucible.
C, Thermo-couple tube.
P, Potentiometer.
B, Storage-battery.
A, Ammeter.
SC, Standard cell.
G, Reflecting galvanometer.
D, Lamp and scale.
X, Adjustable resistance.
L. B. Lamp resistance.
Q, Rheostat.

FIG. 9.—SET-UP FOR TAKING MELTING-POINTS.

Fig. 9 gives a diagrammatic sketch of the set-up required for the taking of melting-points.[15] Two types of potentiometers were used: one a thermo-couple potentiometer made by the Cambridge Scientific Instrument Co., and the other a precision thermo-couple potentiometer made by the Leeds-Northrup Co. The last instrument measures direct to 0.5 microvolt. The reflecting galvanometer was either a Broca instrument, made by the Cambridge Scientific Instrument Co., or a Leeds-Northrup D'Arsonval instrument, type HS. Either instrument is very sensitive to small differences of current. In taking readings the beam of light was brought to zero on the scale at the end of the time-interval and the microvolt reading taken on the potentiometer. The current on the furnace was kept constant by means of the rheostat, which has an adjustable resistance.

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[15] For a description of the potentiometer method of measuring temperatures consult *Principles of Metallurgy*, p. 165 (1910).

IV. THREE SERIES OF SLAGS.

Three series of slag were investigated. In many smelting-problems the iron, silica, and alumina bear practically a constant ratio to each other in the smelting-mixture, for with sulphur and copper these substances make up the bulk of the ores, and the only variable in the smelting-mixture is lime, added as limestone flux. On this assumption three basal or ratio slags were selected, in each of which there is a different ratio of SiO_2 : FeO : Al_2O_3, and to which were added different amounts of lime to make the slags of a given series.

TABLE V.—*Composition of Basal Slags.*

Number of Series.	Composition.				Ratio of SiO_2 : FeO : Al_2O_3.
	SiO_2.	FeO.	Al_2O_3.	CaO.	
	Per Cent.	Per Cent.	Per Cent.	Per Cent.	
I.	42.65	36.12	11.30	6.20	377 : 319 : 100
II.	44.50	38.20	11.85	5.10	376 : 323 : 100
III.	44.30	43.1	4.85	6.80	914 : 888 : 100

Two of the series are relatively high in alumina and one is low in alumina. For a given silicate degree, series I. is lowest in iron as a base and series III. is highest. The slags of the three series therefore furnish data showing the effect of varying silicate degree, and the effect of iron and lime as a base, and also the effect of alumina, within certain limits. The materials for the slags of series I. and II. were the ores and flux of the actual smelting-mixture. These were taken to conform to the actual conditions obtaining at the plant for which a part of this investigation was carried out. Table VI. gives the composition of carefully prepared samples, which were used in making the mixture.

TABLE VI.—*Composition of Material.*

Material.	Composition.								
	SiO_2.	Fe as Fe_2O_3.	Fe.	Al_2O_3.	CaO.	MgO.	Cu.	Loss.	Total.
	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.
Limestone.......	1.94	1.09	......	0.58	52.54	1.09	......	42.49	99.73
Converter-slag.	17.80	73.33	51.29	4.85	1.16	0.47	6.18	......	......
Calcines.........	29.10	30.24	......	5.80	5.44	0.48	16.98	......	......
Tank-slimes.....	51.32	14.08	......	11.24	1.12	......	6.88	......	......

In making the basal slags of the first two series, the above substances were mixed in the proper proportions to yield from 75 to 100 g. of slag of approximately the desired composition. This mixture was carefully smelted down in the electric furnace, in cylindrical crucibles turned from pure Acheson graphite. The fusion was held at a comparatively high temperature for some time, but not high enough to cause any serious "boiling" of the matte, and then allowed to solidify in the furnace. When cold the mass readily drops from the crucible and the matte can be removed from the slag in the form of a button. The slag was then resmelted in the same crucible at a somewhat higher temperature to give opportunity for a second settling of matte, although usually only isolated globules were found after the second smelting. The slag was then crushed and about 20 g. ground fine in an agate mortar, and the metallic iron reduced by the action of the graphite crucible was removed by a magnet. The equivalent of this iron in ferrous oxalate was returned to the slag. This mixture was then placed in a weighed platinum crucible 1.25 in. deep and ⅝ in. in diameter, viz., a narrow high crucible, and slowly melted down into a homogeneous slag. After cooling, the weight of the slag was taken. The melting-point of the slag was then determined by means of the bare thermo-couple, *i.e.*, no protective tube was used. The insulating-tubes reached to within about 0.75 in. of the hot-junction. After the last determination the slag was remelted, the couple removed and the slag allowed to solidify. Then limestone was added either in 0.5-, 1- or 2-g. additions, the crucible replaced in the furnace and the new slag melted down while stirring with a platinum rod. The temperature was then raised until the slag was thinly fluid, the thermo-couple carefully inserted and centered, and after solidification reheated and new melting-points taken. In this manner the points of the whole series were taken consecutively. After the final determination the crucible with its contents was weighed, the slag then remelted until thinly fluid and then poured into water in a porcelain dish to get the sample for the chemical analysis. From the weights of the slag, the limestone taken, and the analysis, the composition of the slags of the series was calculated. This method was adopted because of its neatness and the little wear it caused on platinum, for a whole series could be run without the necessity of cleaning slag from the crucible, except at the end. The possible error in the composition of each slag due to calculating back from the final analysis is probably very small with the weighings carefully made, and with no dusting on the addition of limestone.

For the slags of series III., pure analyzed reagents, alumina (ignited), silica, calcium carbonate, and ferrous oxalate were used to make the basal slag. Otherwise the procedure for the series was the same as for the first two, except that the basal slag was analyzed and the other slags of the series caculated from this analysis and the original weight and the limestone addition. Tables VII., VIII., and IX. give the data on the three series in condensed form.

TABLE VII.—*Composition and Melting-Point of Slags of Series I.*

Number.	Composition.					Silicate Degree.	Melting-Point.		
	SiO_2.	FeO.	CaO.	Al_2O_3.	Total.		Millivolt Readings.	½ Cold-Junction.	Melting-Point.
	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.			° C.	° C.
1.	42.65	36.12	6.20	11.30	96.37	1.40	8.550	9	979
2.	40.66	34.33	10.80	10.70	96.49	1.28	8.815	16	1,011
3.	38.80	32.60	14.95	10.02	96.37	1.19	9.380	10	1,060
4.	36.90	31.10	18.52	9.75	96.27	1.09	9.625	11	1,081
5.	35.40	29.65	22.12	9.35	96.52	1.02	10.250	12	1,137
6.	33.92	28.48	25.14	9.00	96.54	0.96	10.420	14	1,151

Original weight of slag = 20 g. Weight of slag plus additions = 25.751 g. Weight of final slag in crucible = 25.0575 g. Difference = 0.6935 g. on rod and couple.

TABLE VIII.—*Composition and Melting-Point of Slags of Series II.*

Number.	Composition.					Silicate Degree.	Melting-Point.		
	SiO_2.	FeO.	CaO.	Al_2O_3.	Total.		Millivolt Readings.	½ Cold-Junction.	Melting-Point.
	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.			° C.	° C.
1.	44.50	38.20	5.10	11.85	99.65	1.51	9.260	13	1,050
2.	44.00	37.80	6.14	11.73	99.67	1.48	9.000	13	1,025
3.	43.50	37.30	7.15	11.60	99.55	1.45	8.600	13	988
4.	43.20	37.00	8.16	11.51	99.87	1.42	8.600	7	982
5.	42.25	36.20	10.04	11.23	99.72	1.38	8.850	10	1,010
6.	40.50	34.65	13.60	10.75	99.50	1.28	9.300	13	1,054
7.	38.90	33.25	16.80	10.40	99.35	1.20	9.700	14	1,091
8.	37.55	32.10	19.90	10.05	99.60	1.13	9.820	13	1,103
9.	36.20	30.807	22.56	9.67	99.30	1.06	9.840	15	1,107

Original weight of slag = 23.645 g. Weight of slag plus additions = 29.683 g. Weight of final slag in crucible = 28.891 g. Difference = 0.814 on rod and couple.

TABLE IX.—*Composition and Melting-Point of Slags of Series III.*

Number.	Composition.					Silicate Degree.	Melting-Point.		
	SiO_2.	FeO.	Al_2O_3.	CaO.	Total.		Millivolt Readings.	½ Cold-Junction.	Melting-Point.
	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.			° C.	° C.
1.	44.3	43.10	4.85	6.80	99.05	1.66	8.750	10	1,000
2.	42.65	41.51	4.69	10.29	99.14	1.54	8.850	10	1,010
3.	41.11	40.01	4.51	13.53	99.16	1.44	9.000	19	1,034
4.	38.34	37.22	4.21	19.35	99.12	1.27	9.740	15	1,097
5.	35.92	34.97	3.94	24.43	99.26	1.13	10.540	17	1,167
6.	33.79	32.89	3.71	28.92	99.31	1.03	11.000	22	1,207
7.	31.90	31.05	3.50	32.90	99.35	0.934	10.690	26	1,186

Original weight of slag = 14.40 g.

The melting-point curves constructed from the data of the three series are given in Figs. 10, 11, and, 12. In Figs. 11 and 12 the lower curve is the melting-point curve and the upper one is the fluidity-curve. The curves are plotted on the basis of silicate degree and temperature, a given silicate degree representing a certain mixture of two minerals, one an ortho-silicate and the other a meta-silicate. The percentage of lime in the slags is also indicated. The preceding tables give the composition of the slags.

From the appearance of the curves, the series fall into a type in which a eutectic, or mixture of a lowest melting-point, occurs. While no petrographic examination was made of the slags of these series, they are of similar composition and type as the reverberatory-slags discussed in Section II. of the present paper. There seems but little doubt that series III. is an augite-olivine one, while series I. and II. are perhaps the same, but more probably augite-melilite, considering the amount of alumina present. The silicate degree of the eutectic of series

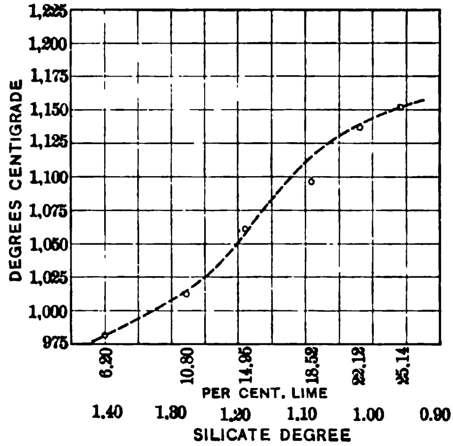

FIG. 10.—MELTING-POINT CURVE, FIRST SERIES.

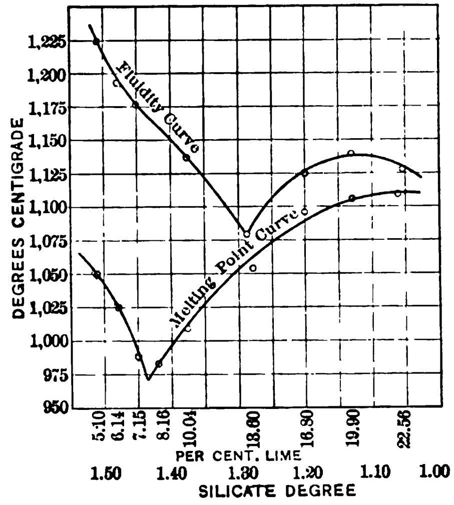

FIG. 11.—MELTING-POINT CURVE, FLUIDITY CURVE, SECOND SERIES.

III. falls at about 1.62, which is close to what J. H. L. Vogt states it to be for a certain augite-olivine series. The silicate degree of the eutectic of series II. falls at about 1.44, which corresponds to what it should be approximately for a certain augite-melilite series, according to the same authority. It will be noted that the total of the constituents of the slags in series I. adds up to but 96.5 per cent. It is probable that in these slags some magnetite is still present, which has the effect of increasing the silicate degree above that calculated. This will displace the curve to the left in Fig. 9. That this should be

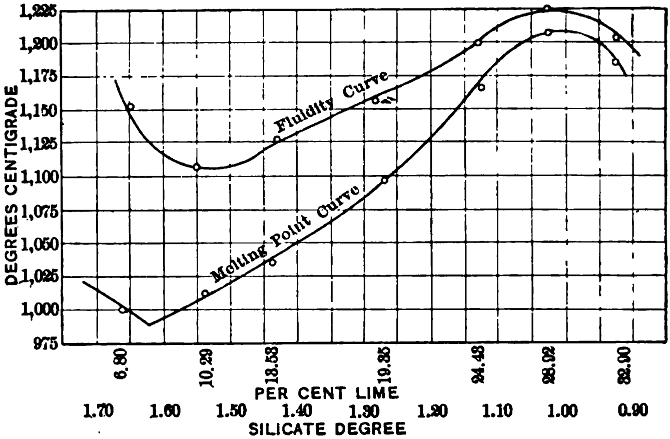

FIG. 12.—MELTING-POINT CURVE, FLUIDITY CURVE, THIRD SERIES.

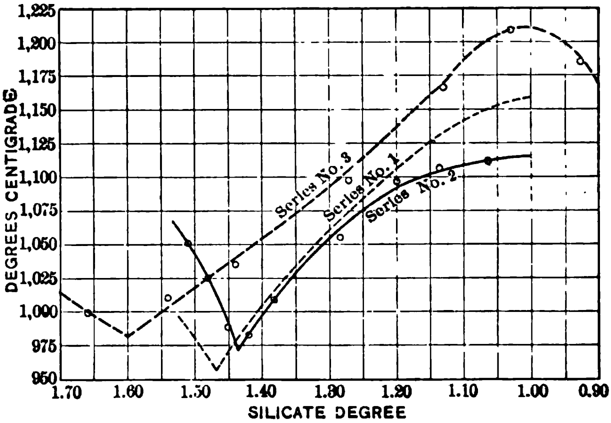

FIG. 13.—MELTING-POINT CURVES.

done is perhaps apparent when it is noted that no maximum is reached in this curve at a silicate degree of 1, as is done in series II. and III. A maximum is expected at this point in a series of this type, for here occurs one of the end-members of the series, viz., either olivine (lime-iron olivine) or melilite.

The other maximum occurs at a silicate degree of 2, or the augite end. It is also very probable that in series I., the first one to be worked out, the eutectic was missed, and is located between slags 1 and 2. That this is so is perhaps apparent when series II. is considered, which was worked out with much smaller addition of limestone in this region. On this assumption the melting-point curve for series I. is replotted in dotted lines in Fig. 13, with the melting-point curves of the other two series.

Discussion of Results.—From an inspection of the curves in Fig. 13 and the composition of the slags given in the preceding tables, it is apparent that the slags of lowest melting-points lie roughly between a silicate degree of 1.4 and 1.65, dependent upon the base composition. This confirms what has long been the opinion of copper-metallurgists. With a given silicate degree, within the range of 1.70 to 1.00, the replacement of iron and lime by some alumina, up to 11 per cent., lowers the melting-point, as is apparent when the curves of series I. and II. are compared with the curve of series III. In a given slag of certain silicate degree within the range specified, the replacement of iron by lime raises the melting-point somewhat. This, however, does not mean that a pure ferrous bisilicate, for instance, has a lower melting-point than one in which some iron is replaced by lime, for this condition would be expressed by the series hypersthene-augite, in which a eutectic may also occur. It has reference to series of minerals in each of which lime is necessarily present, as augite-melilite. Then of two slags each of the same silicate degree, the one with relatively more lime has the higher melting-point. The slags investigated have melting-points that lie between 970^c and 1,220° C. The slags of higher silicate degree have comparatively low melting-points, and that portion of the curve to the left of the eutectic, towards the augite end, rises very gradually. For example, slag No. 16 (Table III.), augite with some magnetite, practically a bisilicate, has a melting-point of but 1,037° C. It is, however, probable that the presence of magnetite lowers the melting-point of this slag somewhat.

When, however, the fluidity-curves are examined in connection with the melting-point curves it will be noted that the gap between the two curves widens rapidly as the silicate degree increases, and while from 25° to 40° of superheating is sufficient in the vicinity of the ortho-silicate, from 150 to 175° is required to superheat to fluidity at a silicate degree of 1.5. By consulting Figs. 11 (high alumina) and 12 (low alumina) it may be seen that increased alumina accentuates the gap greatly, and the more highly aluminous slags, due to their great viscosity, require much superheating above the melting-point in order to obtain enough fluidity. It is this phenomenon which makes high-silica, high-alumina slags so difficult to work. As already noted, the minerals orthoclase and albite, high in alumina and silica, are extremely viscous.

Slags approaching near the bisilicate are looked upon with suspicion for reverberatory-furnace work particularly. Of late, however, the incentive to make the more highly siliceous slags has been great. On the score of formation-temperature and melting-point nothing can be urged against these slags, but they do require much superheating, especially if alumina rises above 5 or 6 per cent. There seems to be no good reason why they could not be made in furnaces of ample temperature reserve, such as the present oil-fired furnaces.

In conclusion, I desire to express my cordial acknowledgment for assistance rendered in various ways by Prof. M. F. Coolbaugh, W. A. Coursen, and Floyd Brooks. I am permitted to say that part of the work described was carried out for the Steptoe Valley Smelting & Mining Co. S. S. Sörensen kindly furnished most of the samples used, for which, and for suggestions made, I express my appreciation.

[TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS.]

Recent Developments in the Inspection of Steel Rails.

BY ROBERT W. HUNT, CHICAGO, ILL.

(Cleveland Meeting, October, 1912.)

PERHAPS of all the scientific economic questions which have been claiming the attention of capitalists, metallurgists, manufacturers, directors of public utilities, and the general public of America, during the last few years, none have been of more importance than the one relating to the wear and safety of steel rails.

The discovery of the Bessemer process made possible modern railway development. In that growth, as has generally happened in all progressive movements, all its department have not remained equally balanced. Iron rails permitted the construction of roadways for the passage of power-driven vehicles. In time the speed and weight of such carriages proved too much for the wearing-qualities of the rails, and the limit of railway capacity seemed to have been reached. Bessemer made his invention, which was soon perfected and developed, especially as related to steel rails, and thereby increased the limit of capacity, and vastly greater railway progress was made possible. The Siemens-Martin open-hearth process followed, and was later supplemented by the Thomas-Gilchrist basic improvement.

For a time the Bessemer steel rails of commerce were more than equal to all that was asked of them, but this did not long continue, as evidenced by the records in our *Transactions*. Revolutionary changes occurred in the industrial world, and in no part of it more than in steel-rail manufacture. Great and powerful commercial combinations were made, whose financial resources made possible the supplanting of the then most efficient blast-furnaces, steel-works, and mills, by others of theretofore unimagined capacity. Old traditions and practices of steel-making were wiped away, and, in too many cases, output was apparently the chief aim, with the result that much dissatisfaction with the wear and safety of rails was felt by railway officials, and the general public was concerned as to its safety when traveling. The climatic conditions of the winter of 1911–12, extending over practically the whole of North America, were abnormally severe. A number of accidents occurred through which there was intense suffering and appalling loss of human life. Government investigation threatened to be followed by government control.

The question of chemical and physical specifications governing the manufacture of rails, and the form and weight of the sections, had already been given, and continued to receive, much attention from committees of scientific societies, individual engineers, and metallurgists. There was much difference of opinion as to the proper corrective action, but a consensus of opinion that something should be done, and that quickly. Several conferences between representative railway and steel company officials were held, during which it was made manifest that the desire existed on both sides to treat the question fairly and make honest efforts to reach the best solution of the difficulties. There were millions of dollars involved, but far beyond that came the protection of human life; and something more than conference resolutions was required.

It had long been the general custom for railways to have their rails inspected by their own representatives before their delivery to them by the manufacturers. The extent of this inspection varied with different roads, but it was generally limited to an examination of the rails after they were finished and offered for delivery. This embraced inspection as to accuracy of section, of length, squareness of sawing, correctness of drilling, straightness in line and surface, and freedom from visible physical defects. Most rail contracts, through specifications, provided for chemical composition, physical treatment of the ingots, a limitation of finishing-temperature, and physical testing of representative pieces of the rails; but the accurate carrying out of such details was largely left to the employees of the manufacturers, although practically all contracts specified that inspectors representing the purchaser should have free access to the works of the seller while the rails were being manufactured.

The foregoing outlines the situation as it was in the early part of 1912, when the chief executive officers of several of the principal American railway systems decided to endeavor to obtain better results under their current specifications and rail-sections; or at least endeavor to secure accurate data upon which to determine the actual cause of so many rail-failures.

They reasoned that, as the majority of their rails did not break in service, and gave reasonably satisfactory wear, it would seem that the failures of the lesser percentage could not be entirely due to faultily proportioned or too light sections, poor road-bed, too heavy wheel-loads, or excessive speed of traffic. They, therefore, arranged to try a more thorough system of inspection, by placing inspectors in each manufacturing department of the rail-plants, to be on duty during all working-hours. It was realized that the manufacturers could not be expected to permit the inspectors to dictate how the details of operation should be conducted, or to interfere with the men in their work; but they were aware that the workmen were paid for work on a tonnage basis, which, of course, had the effect, as it was intended it should, of making each and every man anxious to produce all the output possible, and, moreover, that there was not any effectual way of connecting the quality of the labor but of a few of the workmen with individual rails. Therefore, the instructions to the inspectors were to observe carefully all of the manufacturing operations in the department of the works to which they were assigned, and, if they saw any serious variation from the details of the specifications under which the rails were contracted to be made, or from the recognized good practice of that particular plant, they were at once to call the attention of the foreman in charge of that department to what was happening, make a record of the fact, and also report it to the inspector in the succeeding department to which the ingot, bloom, or rail would pass in the progress of its making. This knowledge would cause extra care on the part of the inspectors in the observation of the behavior of the metal in the rolls, at the drop-test, and the most careful examination of the finished rails for defects, assuming that the heat of metal from which they were made had successfully passed the prescribed physical tests; and another important point was to be certain that the rails were correctly marked as to their heat-numbers and respective locations in the ingots from which they were rolled, and that the pieces of rail for drop-testing were accurately taken. The particulars of the inspectors' observations were to be reported in writing to the proper officer of the purchaser, and in any case of neglect of duty or bad work, the fact was to be given in writing to the official in charge of the works, thereby affording an opportunity for immediate investigation, and so be a check on the judgment and fairness of the inspector making the report.

It was believed that the moral effect on the workmen of knowing that their individual responsibility and association with a heat of steel, and the rails made from it, did not cease with its passage from the department in which they were employed, or being shipped from the works, would tend towards their more careful, and, therefore, better, work.

The engineering staff of all large railway systems keep records of the results obtained from the rails used, and if possessing such data as would come from the foregoing described plan, it would be possible to trace the creative history of rails, should they have failed, or, better still, given satisfactory service.

When the plan of the greater inspection was presented to the rail-makers, they, with scarcely an exception, accepted it cheerfully, and why not? They desired to produce good rails, and wanted to do so with profit to their stock-holders. They paid their workmen for faithful and intelligent service, and, in the long run, it is always more economical to do things right than wrong. By the placing of extra inspectors in their works, they were given that many additional monitors over their men without cost to their companies.

The plan calls for placing men on day- and night-service in the steel-producing departments; in the blooming-mill; in the rail-mill, and in charge of the drop-testing; also from 4 to 5 men on the finished rails; making a total of from 12 to 15 inpectors at each plant. This inspection is now used by 33 railway systems, operating more than 122,000 miles of track, and up to Oct. 1,1912, more than 650,000 tons of rails have been shipped. The plan is working with but little friction, and is proving so satisfactory that it will undoubtedly be adopted by other railway systems. It places the companies using the plan in the strong position of using their best efforts to protect the traveling public.

Naturally, the inauguration of so much detail and the assembling of so large a corps of men, as each and all of the 15 rail-mills of the United States and Canada had to be covered, involved much labor and some change in the personnel of the corps, and alterations of and additions to the forms used in making the reports. I append as a matter of information and record a complete set of the forms now being used in the work.

Robert W. Hunt & Co.
1121 Rookery, Chicago, Ill.

SPECIAL INSPECTION REPORT RAIL MILL. (Size of original form, 5½ by 8½ in.)

From ______ Mill. ______ between ______ Date and Hour. ______ to ______ Date and Hour. ______ Report No. ______

Rails for ______ Railroad. ______ Inspector.

No. blooms per in to ______ Size ______ No. rails per bloom ______ No. passes in blooming mill. ______

Heat No.		Time.		No. Ingots Bloomed.	Remarks on Heating and Blooming.	Per Cent. Discard.		Time Charged to Reheat Furnace.	Remarks.
		Charged.	Rolled.			Top.	Bottom.		
1	(15 lines, each numbered consecutively.)								
2									

Use other side for full details. Under "Remarks" show heats reported to rail mill inspectors. Send this report to general office promptly.

Robert W. Hunt & Co.
1121 Rookery, Chicago, Ill.

SPECIAL INSPECTION REPORT RAIL MILL. (Size of original form, 5½ by 8½ in.)

From ______ Mill. ______ between ______ Date and Hour. ______ to ______ Date and Hour. ______ Report No. ______

Rails for ______ Railroad. ______ Inspector.

Section ______

Use other side for full details—Send this report to general office promptly.

Heat No.		Time Rolled.	No. Rails Rolled.	Template Fit.	Condition of Section.	Maximum Shrinkage.	Per Cent. Discard.		Condition of Sawing, Stamping, Lettering, Branding.	Hot Trial Weight.	Condition Rails Leaving Hot Beds.
							Top.	Bottom.			
1	(15 lines, each numbered consecutively.)										
2											

Fig. 1.—Forms for Inspection of Bessemer Steel.

ROBERT W. HUNT & Co.
1121 Rookery, Chicago, Ill.

SPECIAL INSPECTION REPORT BESSEMER STEEL. (Size of original form 8½ by 11 in.)

From ______ Mill. Between ______ Date and hour. To ______ Date and hour. Report No. ______

Rail steel for ______ Railroad. ______ Inspector.

Give full details on other side.

Report only Heats made on this Order—send this Report to General Office Only.

Average time blowing heats ______ No. of converters used ______ size of nozzle ______

Was spiegel added to vessel or ladle ______ Other additions made to ladle ______

Average time ingots held in molds before stripping ______ Additions to molds when teeming ______

Treatment of ingot tops ______ Treatment of molds and stools ______

Size of ingots ______ Nominal No. ingots per heat ______ Average ingot weight heats cast ______

	Heat No.	Condition and Temperature of Blowing.	Additions to Metal in Vessel During Blow.	Time Metal Held.			Reaction after Adding Spiegel.	Condition of Teeming.	Condition of Ingots.	Remarks.
				In Vessel after Recarbonizing.	In Intermediate Ladle.	In Teeming Ladle before Pouring.				
1	(30 lines, each numbered consecutively.)									
2										
3										
4										

FIG. 2.—FORM FOR INSPECTION OF BESSEMER STEEL.

ROBERT W. HUNT & Co.
1121 Rookery, Chicago.

SPECIAL INSPECTION REPORT OPEN HEARTH STEEL. (Size of original form 7¾ by 8½ in.)

From ______ Mill. Date ______ Time worked. Heat No. ______

Steel for ______ Railroad. ______ Inspector.

Additions to furnace ______ Condition bath, slag, reaction ______

Additions to ladle ______ Reaction and temperature ______

Time tapped ______ Weight ingots cast ______ Size nozzle ______

Time furnace empty (ladle full) ______ Size of ingots cast ______ Additions to molds ______

Ingot No.	Conditions of			Time.		Remarks.	Ingot No.	Conditions of			Time.		Remarks.
	Molds and Stools.	Teeming.	Ingots.	Cast.	Stripped.			Molds and Stools.	Teeming.	Ingots.	Cast.	Stripped.	
1	(20 lines, each numbered, and space for remarks.)							(20 lines, each numbered, and space for remarks.)					
2													

Send this report promptly to General Office.

ROBERT W. HUNT & Co.
1121 Rookery, Chicago.

SPECIAL INSPECTION REPORT BLOOMING MILL. (Size of original form 7¾ by 8½ in.)

From ______ Mill. Date ______ Time worked. Heat No. ______

Size of blooms ______ No. of rails per Bloom ______ Inspector.

No. passes in blooming mill ______ Inspector.

Ingot No.	Time Charged to Pit.	Charged to		Conditions of Heating.	Time Bloomed.	Condition of Blooming.	No. Blooms Made.	Per Cent Discard.		Condition of Bloom after Shearing.	Time Charged to Reheat Furnace.	Ingot No.	Time Charged to Pit.	Charged to		Condition of Heating.	Time Bloomed.	Condition of Blooming.	No. Blooms Made.	Per Cent. Discard.		Condition of Bloom after Shearing.	Time Charged to Reheat Furnace.
		Pit.	Hole.					Top.	Bottom.					Pit.	Hole.					Top.	Bottom.		
1		(20 li num- space	nes, bered for	each , and re-													(20 li num- space	nes, bered for	each , and re-				

(NOTE.—The heading of the original form, here given for convenience in three sections on this and the opposite page, extends across the top of the form, which is 11 in. high and 22 in. wide. Below this heading 30 horizontal lines are ruled, each bearing a serial number.)

Section 1.

[8]

ROBERT W. HUNT & Co.,
1121 Rookery, Chicago, Ill.,

SPECIAL INSPECTION REPORT OF STEEL RAIL MANUFACTURE.

For____________________ Supplementing our rail report No.________

Rails rolled by____________________ This report No.________

________ Steel rails________ Lbs. per yd.________ Section Date rolled ________

	Heat No.	Date Made.	Size of Heat Cast.	Additions Made to Metal in Ladle	Time Ladle Held Before Casting 1st Ingot.	Additions Made to Metal in Molds.
1						
2						

Section 2.

Size of nozzle used for casting ingots________Exceptions________

Size of ingots cast________Approximate average weight________

Size of blooms________ No. blooms per ingot________ No. rails per bloom________

No. passes given ingots in blooming mill________Exceptions________

No. passes given in rail mill________Total No. passes ingot to rail________

Remarks on Ingot Molds and Stools, Casting Ingots, Ingots, Top and General Conditions.	No. of Ingots Cast.	No. of Butts and Stickers.	Time Between Casting and Stripping Ingots.	Time Between Stripping and Charging Ingots to Pits.	Time Ingots in Pits before Rolling.	No. of Ingots Bloomed.

Remarks.

Section 3.

Remarks on Heating and Blooming.	Approximate Per Cent. Discard.				Time Blooms in Reheating Furnace	No. Rails Rolled.		
	At Shears.		At Saws					
	Top.	Bottom.	Top	Bottom.				
								1
								2

Fig. 4.—Form for Inspection of Steel-Rail Manufacture.

SUBJECT TO REVISION.

[TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS.]

The Concentration of Iron-Ores.

BY N. V. HANSELL, NEW YORK, N. Y.

(Cleveland Meeting, October, 1912.)

I. GENERAL.

THE preparation of low-grade iron-ores by concentration, whether or not followed by an agglomeration of the concentrate, has in the United States only recently been recognized as a metallurgical process of vast economical importance for its iron industry. This importance of the process lies not only in the fact that it makes possible the use of ores heretofore discarded on account of their low iron-content or their too high percentage of elements deleterious in the iron industry, but also in its production of a raw material for the blast-furnace that is more uniform in its physical character and chemically purer, with a higher iron-content, than the general run of natural ores. When mixed in the burden of the furnace, it lowers the amount of coke and fluxing-stone required per ton of pig-iron produced. The furnace-operations become more regular and more easily controlled. The pig-iron is of higher quality and more uniform in character. In brief, it is becoming recognized that the beneficiating of the natural iron-ore, even if it makes the furnace-ore higher in cost per unit of iron entering into the burden, may mean a lowering of the production-cost of the pig-iron and an improved product.

At the present time, the Lake Superior district furnishes about 80 per cent. of all the iron-ore yearly produced in the United States. If the Southern States are exempted the percentage runs as high as 93 per cent. These ores are showing a steady decrease in grade from year to year. For the past 10 or 15 years this falling off in the average grade has been about 0.5 per cent. a year. The lowered grade of ore means an increased consumption of ore, fuel, and flux per ton of pig-iron, which in its turn causes an increase in labor, relining, and overhead costs (per ton of pig), due to decreased output per day. Writing of Pittsburg conditions as to grade of flux and coke, Edwin C. Eckel[1] says in a recent paper that a decrease of 5 per cent. in the iron-content of the ore involves the additional use of about 0.17 ton of ore, 0.12 ton of coke, and 0.36 ton of fluxing-stone per ton of pig-iron produced. Using these data and inserting approximate current prices for the three raw materials, it will be found that a drop in ore-grade from 50 to 45 per cent. brings with it an increase in cost, for raw materials alone, of about $1.25 per ton of pig-iron. When the other elements of cost, which also increase as the ore-grade lowers, are considered, the total increase becomes still more serious. Mr. Eckel draws the conclusion that it seems safe to assume that the cost of pig-iron will increase at the rate of between 25 and 50 cents per ton for each decrease of 1 per cent. in the average iron-content of the ores to be used.

Considering these facts, notwithstanding the increasing tonnage of fairly high-grade ores that is being imported, it seems reasonable to presume that the demand for high-grade concentrates to be used as an intermixture with low-grade ores in preparing the furnace-burden soon will make itself apparent in a willingness on the part of the ore-consumers to pay the higher price that the concentrated product must command on account of the cost of its preparation beyond the mere mining-cost. The higher cost per unit of iron in the burden will be counterbalanced by the lowering of the cost of the furnace-operations, the increased output of the furnace, and the higher quality of the pig-iron produced.

This broader aspect of the problem gives us the right viewpoint from which to consider the advisability of purifying an iron-ore before charging it into the blast-furnace rather than during the smelting-operation itself.

The object of concentrating ores is generally to enrich their metallic content by reducing the percentage of gangue-minerals inherent in the natural ore. With iron-ores, this may not always be the chief purpose, as sometimes it is found advisable to concentrate ores with an iron-content sufficiently high for ordinary blast-furnace practice, the reason for the treatment being the necessity of reducing the percentage of elements generally objectionable for the iron industry, such as sulphur, phosphorus, copper, titanium, etc. The enrichment of the iron-content is then merely incidental.

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[1] Iron Ore Reserves, *Engineering Magazine*, vol. xliii., No. 6, p. 835 (Sept., 1912).

Not all iron-ores are amenable to concentration in the present state of the art unless there are included under this general heading such metallurgical processes as the roasting of siderites or the nodulizing of Cuban limonites, both of which processes increase or concentrate the iron-content, and possibly also so-called metallizing processes—still in their experimental state—by which it has been claimed it is possible to handle those low-grade, close-grained, hard hematites of the Ranges with the silica so closely combined with the iron-mineral as to be significantly described as "silica poisoned," that to this date have defied all usual methods of concentration. Excluding in our present discussion such special processes, we may say that the basic principles of iron-ore concentration are well recognized and its limitations well known.

The various methods, then, which are at present commercially used in the United States in the concentration of iron-ores, can be classified as follows:

1. Washing of brown, clayey ores and sandy hematites, usually without previous crushing.
2. Wet-separation on jigs and tables of hard hematites, after crushing and sizing.
3. Dry-separation of hematites.
4. Magnetic separation of magnetites (and in a few cases mixed magnetites and hematites) by either dry or wet methods.

My intention in the present paper is not to describe in detail the different processes and the equipment used in the various plants in the United States, but to try to show more broadly the present status of the art and the possibilities for further development.

As can be concluded from the classification of the processes, the choice of method depends almost entirely on the character of the crude ore. The degree of refinement to which the preparation of the ore is carried in each separate case is influenced by such other factors as the comparative merchantable value of a higher or lower grade of product and its relation to the cost of production; the distance from, and the nature of, the market; the possibilities of selling the tailings-product or of obtaining from it some valuable by-product, like copper or apatite, or else the cost of its disposal; the handling of the water, if a wet method is selected, etc. The type and character of the ore is, however, the main consideration. In contemplating this, it is soon found that it is seldom that ores from different deposits are entirely alike in physical respects. A method found very successful for the beneficiating of one ore has to be considerably modified in order to give satisfaction in the treatment of another ore apparently quite similar.

A few remarks on the proper method of investigating the ore in connection with the determination of the method of enriching it are therefore pertinent before an attempt is made to describe the various methods as they have been worked out in the United States. Preceding this description of the difference between the processes, some attention will also be given to recent developments in ore-crushing, which enters as an important part of every installation of this kind, with the possible exception of washing processes for clayey ores.

As in every ore-concentration problem, the first step towards arriving at the best method to use is a careful investigation of the ore in the field and in the laboratory, followed by practical crushing- and concentrating-tests in the testing-plant. The general structure of the ore is studied, the mineral form of the metal-bearing constituents and the gangue-minerals is determined, as is also the size of the mineral grains, and the way they are attached to each other. Before a separation of the various grains can be made, they have to be freed from each other by crushing or otherwise, so that the difference in their physical qualities—specific gravity, magnetic permeability, etc.—can be taken advantage of in their separation. Valuable information for the determination of the most efficient and economical method for the comminution of the ore to the right degree of fineness for the separation can be procured in these preliminary investigations by a microscopic study of characteristic thin sections. It gives a correct knowledge of the mineralogical composition of the ore, which otherwise may be difficult to obtain in finely granulated and closely intermixed ore-samples. It will, for instance, give an explanation of the high iron-content in the tailings-product of an ore with gangue-minerals consisting of iron silicates high in iron. The percentage of iron in such minerals varies greatly in specimens from different localities, as investigations by Prof. W. Petersson[2] indicate. In amphibole, for instance, he found from 2.11 to 16.50, with an average in samples from nine sources of 8.76 per cent.; in pyroxene, from 3.05 to 12.00, with an average in six samples of 9.2 per cent.; in garnet, from 17.45 to 21.50 per cent., and so on.

Again, the study of the microsection may indicate that there is an appreciable amount of hematite intermixed with the magnetite, making the iron-recovery by magnetic separation unsatisfactory. It will show how the ilmenite and magnetite occur in a titaniferous ore and what may be the possibility of separating them. It will clearly establish how the sulphur- and phosphorus-carrying minerals are intermixed with the rest of the aggregate, and thereby give data helpful in determining how fine to crush the ore in order to make possible their elimination in the separation.

Figs. 1, 2, and 3 show photographs of three such thin sections enlarged 16 times. In these the opaque or black parts are the magnetite; the more or less grayish, various iron silicates; and the white, chiefly quartz.

Fig. 1 is a coarsely crystalline gneiss carrying magnetite from an ore-occurrence in the northern Adirondacks, N. Y. The minerals are all in allotriomorphic grains, with quartz and pyroxene forming the largest part of the sample. The magnetite is in grains of all shapes and sizes, and forms, seemingly, the cementing-material between the rest of the aggregate. A crushing to ⅛ in. will permit a clean separation.

Fig. 2 illustrates a bedded magnetite consisting of alternating layers with either quartz or magnetite predominating. The magnetite occurs in well-developed crystals, and, although the ore has to be ground fine for separation, it will give clean, high-grade products. The sample is from an eastern Mesabi deposit.

Fig. 3 is a sample of an amorphous hematite bedded alternately with micro-crystalline quartz. The sample is from eastern Vermillion and characteristic of several deposits there.

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[2] *Jern-Kontorets Annaler*, vol. lviii., p. 359 (1903).

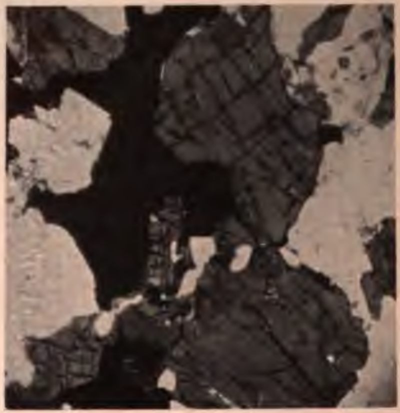

FIG. 1.—CRYSTALLINE GNEISS AND MAGNETITE.

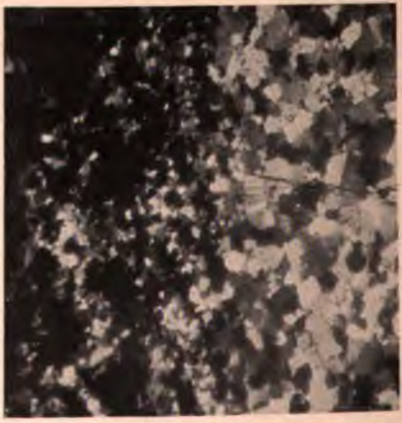

FIG. 2.—QUARTZ AND MAGNETITE.

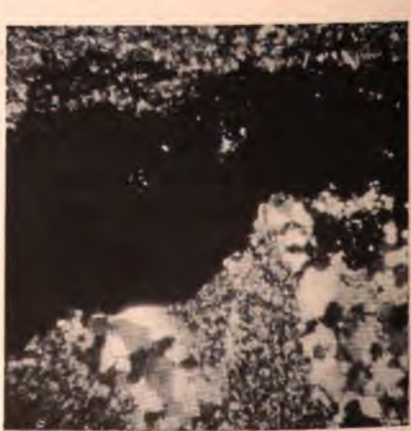

FIG. 3.—QUARTZ AND HEMATITE.

They crush more than 1,000 tons per hr. The percentage of idle time would therefore be considerable, which would make the crushing-cost high, inasmuch as they consume a large amount of power, whether running light or operated to their full capacity.

Regardless of the type of crusher that has been selected as the initial crushing-apparatus of the plant, it is evident that it ought to be of sufficiently large size to take the largest piece that can be economically loaded and brought to it. It has opened the way for steam-shovel loading in open quarries, with its many contingent economies. Block-holing is eliminated. The stopes can be carried higher. The drilling- and blasting-expense is greatly reduced. It also creates a greater independence of labor-conditions by considerably decreasing the number of men needed in the quarry.

The rule is, then, that the first crusher must take any stone that the shovel can load. For a jaw-crusher this means a receiving opening of at least 48 by 60 in. in size. Some recent installations are using sizes ranging up to 60 by 84 in., an enormous machine that can take almost anything that the largest steam-shovel can lift into a car.

The preliminary crusher usually breaks the ore to a size of about 8 in. It is followed by secondary crushers of either type, screens, and other machinery; the crushing-plant being laid out as an ordinary stone-crushing plant with a gradual reduction of the ore to 1.5 in. To this point, the handling of all kinds of hard ore is alike, regardless of the method that has been selected for the subsequent treatment, based on the study of the ore itself and the results obtained in tests.

In this subsequent treatment, the concentration is generally designated as coarse or fine, two relative terms indicating whether the particles to be separated in the final stage of the process are coarse as sand or gravel or fine as slime. Whenever it is possible, coarse concentration is preferable, as it represents economy. As will be shown later, it is often combined with fine concentration, by which the middlings and tailings obtained by it are re-treated.

These general remarks appertain to the concentration of any low-grade iron-ore, although only more indirectly to the washing of brown ores or sandy hematites, in which process a crushing of the crude ore is not generally required.

III. WASHING.

The washing of brown ores and sandy hematites has been employed for many years in the South: in Georgia, Virginia, Alabama and elsewhere. Taking Georgia as an instance: the ore is found in the form of boulders and nodules in residual clays, and the proportion it bears to the entire material varies greatly. The concentrating process is simple and the plants in the South do not often involve any large outlay of money. A log-washer followed by shaking-screens or trommels, with picking-belts for the oversize and jigs for the fines, constitutes the usual equipment of the washing-plant. The slime is generally wasted without any attempt being made to recover its iron-content. The average iron-content of the concentrate is probably between 40 and 46 per cent., although at places it may run as high as 56 per cent. Generally, from 3 to 4 tons of wash material is handled for 1 ton of clean washed ore.

The peculiar development of the iron industry of the South makes the process well-suited to the conditions. The iron-ores are abundant and cheap, and the blast-furnaces are generally placed in close proximity to the mines. The ores often contain a considerable amount of lime with the silica, so that there is no great incentive to make a product of very high iron-content. The proper field for improvements in the process lies therefore in recovering part of the iron which now is wasted with the slime and the fines. It is reported that tests are being made with this point in view with reciprocating-tables. Experiments are also being carried on at several places with magnetic separators.

On the western Mesabi an experimental plant along similar lines was erected by the Oliver Iron Mining Co. in 1907, to determine the most economical way of treating the low-grade sandy hematites of the district. These lean ores occur as layers of hard hematite and free sand, alternating with paint-rock and taconite in varying amount. The results of the experiments were so satisfactory that a couple of years later a large permanent washery was built, which began working in the spring of 1910, and has since been in continuous operation. The plant consists of a number of parallel units, each with a capacity of about 200 tons of crude ore per hour. The principle of the treatment is the same as in the small washeries of the South, but the concentration is carried further, inasmuch as the overflow from the log-washers is concentrated on Overstrom tables for the recovery of a part of its iron-content. No crushing is done, because most of the large pieces are either good ore or taconite, which is picked out on the picking-belts. A similar plant has recently been put in operation by the Wisconsin Steel Co. at Nashwauk, Minn., to serve the Hawkins mine on the Mesabi range. It also is built on the "unit" system, each unit consisting of a grizzly with bars spaced 9 in. apart, a trommel with 2-in. perforations, a picking-belt for the oversize from the trommel, and two log-washers, for the undersize. The overflow from the log-washers is thickened in settling-tanks, and re-treated in auxiliary washers. No tables are employed. The present total capacity of the plant is about 4,000 tons daily.

The introduction of this system of washing ore on the Mesabi range is making it possible to utilize profitably the vast deposits of low-grade sandy iron-ore found in various parts of the range. This ore as it comes from the mines runs from 35 to 50 per cent. The iron-content of the concentrate is from 58 to 62 per cent.

IV. JIGGING.

Low-grade specular hematites, in at least two places in Canada, are concentrated on jigs after having been crushed and screened along the usual lines employed in the wet method of concentrating ores by gravity. No attempt is made to produce a high-quality product, the object of the treatment being to reduce the silica and obtain an ore of a guaranteed iron-content with a minimum amount of fines. The crushing is therefore not carried further than to about 1.5-in. pieces. A drawback is the relatively large percentage of iron in the waste, which is unavoidable on account of there being no recrushing of jig-tailings. The recovery of this iron would require a fine crushing of the tailings, with a separation on sand-jigs and tables. The resultant product would be fine concentrates that would probably have to be agglomerated before shipment.

The concentration of low-grade hematites and brown ores by jigging will undoubtedly be more used in the future. It is only relatively recently that large jigs have been constructed that can handle ores crushed only to 1.5 or 2 in. in size.

An interesting development in this connection is the Woodbury system of concentration, which is used extensively in the mills of the Lake Superior copper country, and which seems well-suited for the treatment of low-grade hematites. The ore is fed unsized to a plunger-jig so arranged that the flow of ore travels across the jig together with the water-currents caused by the plunger-impulse. The jig acts as a classifier, and delivers, besides coarse concentrates, hutch-sand, that may be re-treated on tables, middling for regrinding, and slime, approximately 60-mesh and finer, for subsequent treatment on tables. The 24-in. classifying-jig has a stated capacity of 250 tons per 24 hr. of unclassified ore, crushed small enough to pass ⅜-in. round hole. The 48-in. machine treats 500 tons per 24 hr. The classifying-jig is generally followed by several compartments of Woodbury jigs to complete the separation of concentrates, middlings, and tailings. The system greatly simplifies the layout of a mill.

The great capacity of the modern jigging-machinery is an illustration of the rule that nowadays things have to be done in a large scale in order to be done effectively and profitably.

V. Dry Concentration.

Before passing to the description of the magnetic concentration of iron-ores, a few words ought to be said about the dry process for the enrichment of "non-magnetic" iron-ores. There is, in the first place, the straight drying process as used at the Hollister mine in the Crystal Falls district, Mich., and at the Brunt mine on the Mesabi range, the purpose of which is a reduction in hygroscopic moisture of the ore, chiefly in order to lower the freight-charges. The Hollister ore assays 49 per cent. of iron, net. It contains from 13 to 14 per cent. of hygroscopic moisture. By the drying this moisture is lowered to 6 or 7 per cent. and the iron increased correspondingly.

There is also the E. G. Goltra process, which is developed on a commercial scale at Waukon, Iowa. Letters patent, recently published, by which the process is protected, describe it as follows: "The ore and its gangue are heated at a temperature and for a period sufficient to render the gangue brittle and easily separable from the ore. The ore and its gangue are tumbled to shake loose the gangue from the ore, and the ore and gangue are subjected to a strong air-blast so as to sweep, clean and remove the gangue from the ore." In regard to the practical application of the principle and the economical results obtained nothing has yet been published. It is known, however, that the drying- and calcining-furnace is similar to a nodulizing-furnace with the exhaust-fan at the charging-end. The discharged dry ore is screened. The oversize is cobbed by hand on a picking-belt. A part of the fines is concentrated on air-tables.

The Goltra process is probably best adapted to treat ores containing chiefly clayey gangue-constituents.

Air-tables have a limited use elsewhere in connection with the separation of dried finely-granulated hematites. They correspond to the shaking-tables used in the wet mills, and usually have a similar reciprocating, differential movement. The table-top is made of broad-cloth through which air is pressed either in a steady or a pulsating current.

At the briquetting-plant for flue-dust at Duquesne, Pa., such tables are used to re-treat the non-magnetic product from magnetic separators. They deliver hematite as concentrates and coke and lime-dust as tailings.

VI. Magnetic Separation.

Magnetic separation is applied to iron-ores which are naturally magnetic. It may also be profitably used for such ores as can cheaply be converted into a magnetic state.

The magnetite is strongly magnetic, *i.e.*, highly permeable to the lines of magnetic force. Introduced in a magnetic field it becomes itself a magnet, that is attracted to the magnetic poles from which the lines of force emanate. This causes the separation from the gangue-minerals, which as a rule are influenced only slightly, if at all, by the magnetic flux. The magnetic separation is simple and efficient and is the only method now used for the enrichment of low-grade magnetites.

he hematite is relatively weakly magnetic, *i.e.*, in a magnetic field it concentrates within itself but few lines of force. It can be separated from its gangue-minerals only by being exposed to a very strong and concentrated magnetic field. Machines designed for such weakly magnetic ores, although expensive in first cost and with rather small capacity when compared with separators for magnetite, will probably gradually be developed to commercial usefulness for hematites. What is particularly needed is a separator that can treat efficiently the overflow and slime from log-washers or hematites ground fine in water. It is probable that the evolution lies in this direction rather than in a conversion of the hematite into a magnetic oxide by heating in a reducing atmosphere.

Siderites give off carbonic acid when roasted at a temperature of only 300° C. and form a substance having the approximate composition of Fe_3O_4. They can then be separated as natural magnetic iron-ores. There is undoubtedly a field to cultivate in this direction in many parts of the United States.

The concentration of low-grade magnetites on magnetic separators is used at present to a relatively large extent for converting into a marketable blast-furnace ore the magnetic iron-ores of the East, which seldom occur sufficiently high in iron to be smelted in their crude state. A number of plants are in operation in the States of New York, New Jersey, Pennsylvania, and North Carolina, and in eastern Canada; some of them are well-known in their various details from recent descriptions in technical journals. The different methods employed at these works can advantageously be divided into the following three classes:

1. Coarse crushing with cobbing.
2. Fine crushing with dry separation.
3. Crushing with fine grinding and wet separation.

1. *Cobbing.*—If it has been found that the ore when broken to 1.5 or even to 0.5 in. in size contains a large percentage of reasonably pure magnetite, it can be separated into two products, of which the one may be a saleable furnace-ore. The other is either a waste product so low in its iron-content that it would not pay to fine-crush it in order to recover this iron or a low-grade material that is ground finer and concentrated.

This separation of coarsely-crushed ores is called "cobbing," the magnetic cobber being substituted for the hand cobbing on picking-belts sometimes used for non-magnetic ores.

Cobbing-separators, therefore, like picking-belts, are often used in magnetic separating-plants as an intermediate step for reducing the amount of ore that has to be fine-ground. There are two ways by which this can be accomplished, *i.e.*, either by picking out, as mentioned, the rich ore-pieces by the use of a somewhat weak magnetic field in the cobbers or by the elimination of pieces of rock. In the latter case a stronger field is used so that all pieces containing sufficient iron to make them worth fine-grinding are recovered as the head-product. Sometimes, the plant is laid out with an initial cobber operated with low amperage, producing a rich furnace-ore. This is followed by a secondary machine with a stronger magnetic field by which pure rock pieces are discarded. The rest, being a middlings product, is reduced in size and re-treated.

2. *Dry or Wet Magnetic Separation.*—By the dry method, the aim is to produce as coarse a concentrate as is consistent with a clean separation and a satisfactory recovery. The procedure is therefore gradually to reduce the ore to the right degree of fineness. Each crushing-machine is followed by a screening-device so that undersized particles can be taken out and not subjected to further comminution. Drying the ore is required both on account of the difficulty of screening damp material and because dry separators give good results with clean products only when working on perfectly dry ore. Dampness of the ore causes the finely-crushed particles to stick together. Even with drying, it is not possible entirely to prevent non-magnetic dust from adhering to the magnetic particles, which tends somewhat to lower the iron-content of the concentrate or prevent an efficient reduction of phosphorus. This adhesion can be somewhat lessened by passing a strong current of air through the stream of concentrate as it is being discharged from the separator, by which non-magnetic dust is blown out and by means of an exhaust-fan conducted outside the mill. A close screening of the crushed ore and a feeding of the various sizes to different separators have been shown materially to improve results in dry separation.

As a rule, however, the dry method is particularly adapted to ores of coarsely crystalline character, such, for instance, as the

Mineville ore of New York State. The several separating-plants in operation at Mineville are giving excellent technical and economical results. They constitute, in reality, the best examples of American practice in ore-separation, and serve as models for similar installations elsewhere. A great deal of experimental work has always been carried on there and the system has gradually been brought to a high state of simplicity and efficiency.

At present there are four mills in operation at Mineville, treating ore from the various shafts and deposits. As these mills form the largest dry-concentrating installation in the United States, with the latest and most approved equipment, an outline of the process as practiced in one of the mills will be given.

The No. 3 mill receives the ore from the two shafts of the Harmony mine. The ore is crushed in the two shaft-houses to 4-in. size. It is then delivered to a 300-ton bin, from which it is fed to an elevator. This discharges by means of a dividing-box to two revolving screens with 2-in. holes. The oversize is crushed in secondary crushers and returned until the whole flow of ore is reduced to 1.5-in. size. It then passes over two magnetic drum-cobbers, which remove the richer part as finished product. The rest continues over two cobbers of a different type with stronger field, which eliminate pieces too lean for further treatment; 12 per cent. of iron is considered the approximate limit. The heads from these cobbers are crushed in rolls to $\frac{1}{8}$-in. size and then delivered to Ball & Norton separators. The tailings product is crushed finer and re-separated. The concentrates from all the separators are collected in the shipping-bin and the final tailings are transported to the tailings-dump or sold for concrete-work or ballasting. The capacity of the mill is about 100 tons of crude ore per hour and the power-consumption is about 3 kw-hr. per ton of crude ore.

Analyses.

	For 1 ton of Concentrates.	Iron. Per Cent.	Phosphorus. Per Cent.
Crude ore	1.5	51	0.2
Concentrates	1	64	0.08
Tailings	0.5	10	

At Lyon Mountain, N. Y., is treated the ore generally known under the name of Chateaugay ore. The general system of concentrating is similar to that at Mineville. All the ore is crushed to 0.25 in. before being delivered to the separators, which are of the Ball & Norton double-drum type. Middlings and tailings are recrushed and re-separated.

Analyses.

	For 1 ton of Concentrates.	Iron. Per Cent.	Phosphorus. Per Cent.
Crude ore	2.6	32.0	0.022
Concentrates	1	64.0	0.009
Tailings	1.6	9.0	0.034

A screen-test of the concentrates made a few years ago with full analyses of all screenings has been plotted on the chart,[3] Fig. 4. It clearly shows how improvements in the system could be instituted. By the elimination of the coarse that stays on the 8-mesh sieve, and the very fine that passes through 100-mesh, of which there is no less than 15.6 per cent., the iron-content of the concentrates would be raised to more than 67 per cent. and the phosphorus reduced below 0.008 per cent. It shows that the ore ought to be ground so as to pass ⅛-in. mesh, and it also shows that non-magnetic dust adheres to the fine concentrates, lowering the value of the whole product.

The wet method would unquestionably be technically superior. The whole mass of ore would be crushed in water to, say, 20-mesh. There would be no middlings and tailings to retreat, and no expense for drying the ore. The concentrates, washed free from non-magnetic particles, would show up better in both iron and phosphorus than by the present method.

For finely-granulated ores, such as are illustrated in the thin sections, Figs. 2 and 3, the wet method is probably indispensable. The ore has to be ground to 60-mesh before a separation of the magnetite grains from the gangue can be successfully attempted. Crushing and grinding the ore to such a degree of fineness produces a large amount of absolute fines. To separate these dry is out of the question. To crush dry is possible, but not advisable, as only with great difficulty can the fine dust produced in the operation be kept reasonably well inclosed. It will pass through the finest cracks. Even with strong exhaust-fans connected to casings around the various machines, the dust escapes, fills the atmosphere in the mill and makes the work in the plant disagreeable and unhealthful, besides causing difficulties in the up-keep of the bearings, etc.

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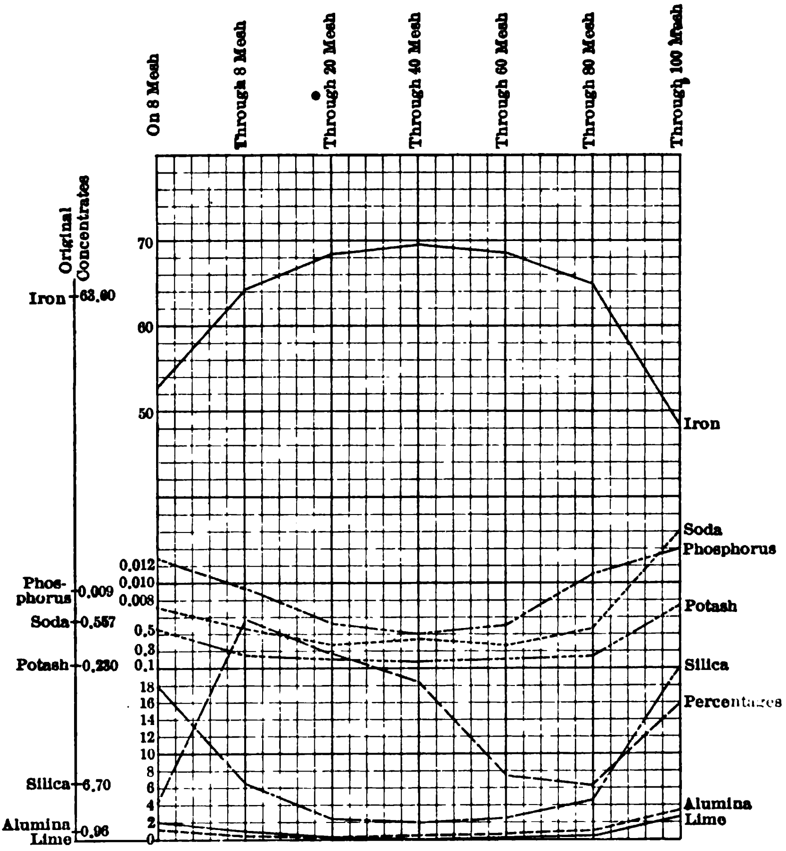
[3] *Engineering Magazine*, vol. xxxviii., No. 4, p. 533 (Jan., 1910).

Grinding in wet ball-mills can be done cheaply, and at the same time permits a very simple layout of the plant. After the preliminary crushing to 1.5-in. size the ore is fed directly to the ball-mill and in one operation ground to 20- or 30-mesh size in water. Driers, rolls, screens, and elevators are unnecessary. The ore, discharged from the ball-mill, is carried by a current of water directly to the separators. With some ores high-grade concentrates are here directly obtained. With other ores a finer grinding is necessary if high-grade



One Scale for Iron, Silica, Alumina, Lime and Percentages
Another Scale for Soda and Potash
A Third for Phosphorus

FIG. 4.—SCREEN-TEST OF IRON-ORE CONCENTRATES AT LYON MOUNTAIN, N. Y.

concentrates are desired. The proper method, then, is to use after the ball-mills magnetic separators of such intensity that they will pick up the greater part of the iron mineral, leaving a tailings-product that can be wasted. The concentrate is crushed finer in pebble-mills and re-separated, yielding a product of any desired quality.

Among wet concentrating-plants for iron-ores in America are to be mentioned the one at Lebanon, Pa., operated by Pennsylvania Steel Co., on low-grade iron-ores from Cornwall Ore Banks, Pa., and the one at Sellwood, Ont., belonging to Moose Mountain, Ltd. At both places the Gröndal magnetic separators are employed. At the Moose Mountain plant, which is very modern in layout, having only recently been completed, the ore from the quarry is broken in a 36 by 48 jaw-crusher to about 3-in. size; it is then distributed to a four-unit plant, each unit consisting of one Jordan impact crusher, which is supposed to comminute the ore to less than 30-mesh, two Gröndal separators, one Hardinge conical pebble-mill, finishing-separators, and dewatering-machinery. The concentrate is formed into briquettes by a special type of briquetting-machine developed by Moose Mountain, Ltd., and is sintered in Gröndal briquetting-kilns.

Analyses.

	For 1 ton of Concentrates.	Iron. Per Cent.	Phosphorus. Per Cent.
Crude ore	2	36.19	0.072
Concentrates	1	65.58	0.020
Tailings	1	6.0	

In this short review of the present status of development of the magnetic concentration of iron-ores in the United States and Canada, only a few of the mines have been named at which separating-plants are in operation. The total capacity of all the magnetic separating-plants in America exceeds considerably 2,000,000 tons of concentrated ore yearly.

In certain parts of Europe, particularly in Sweden and Norway, magnetic ore-separation is used extensively. Norway prides itself on having the largest single plant of this kind. It is at the Sydvaranger mines in northern Norway. The ore there is similar to that of Moose Mountain. It is concentrated and briquetted by the Gröndal method: The present yearly capacity of the plant is 500,000 tons of concentrates, and it is now being enlarged to a capacity of 750,000 tons a year.

3. *Concentrates.*—The concentrates from the separating-process can be used directly in the blast-furnace without any agglomeration, unless the material is so fine that its use would cause inconvenience. In some cases, as at Lyon Mountain, N. Y., and Benson Mines, N. Y., the concentrates are classified either by screening or by being exposed to a strong air-current. The fines so produced are agglomerated in nodulizing-furnaces. The coarser material, which is supposed to be free from grains finer than 40-mesh, can be marketed and is used without any disadvantage.

For the concentrates from wet-separating plants, which generally are much finer than those from dry plants, agglomeration is usually required before the product can be marketed. There are three methods for the agglomeration of such fine ore that are used in America.

1. Nodulizing in revolving cylindrical kilns.
2. Blast-roasting by the Dwight-Lloyd or the Greenawalt process.
3. Briquetting by the Gröndal method.

In my paper[4] read at the New York meeting, February, 1912 a comparison between the various processes is given. They all convert the fine ore into a lumpy condition suitable for charging in blast-furnaces.

VII. Conclusions.

In making the first rough estimates on the probable revenue in the exploitation of a low-grade concentrating iron-ore, proper attention must necessarily be given to the tonnage of ore available and to the cost of mining it. Generally, any low-grade proposition requires the investment of a large capital in a concentrating-plant and in many cases in an agglomeration-plant, indispensable for the conversion of the fine concentrates into a marketable product. The operations have therefore to be laid out in a large scale, because otherwise the fixed charges distributed on the tonnage will make the cost of production seem prohibitive. A large tonnage of ore in sight, insuring operations on a large scale for a number of years, is therefore a prerequisite.

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[4] *Bulletin No.* 65, May, 1912, pp. 557 to 574.

In the case of iron-ore, it is generally not difficult to predetermine the available supply of ore, and with magnetic ores it is even easier than with hematites, because the magnetic qualities of the ore make it convenient to employ some method of magneto-metrically surveying the ore-deposit. If correctly done, this will give the extent in field of the ore-deposit, together with useful data on the dip and pitch. By a few diamond-drill holes, where the magneto-metric map indicates the best location for them, the quality of the ore in depth and its continuity are determined.

With these preliminaries well done, the development of the deposit can be quite accurately planned. The cost of the various operations can be estimated very closely from figures obtained elsewhere in similar undertakings, and modified to suit the local conditions as experience dictates. The risk generally considered inherent to mining-enterprises is almost entirely absent. . The consumption of iron-ores is large and as a rule increasing from year to year. The American production of pig-iron during the last 100 years has on the average somewhat more than doubled in each decade. The market is therefore well established. The market price of the ore has varied considerably and has, the last couple of years, been very low. It is, however, at present improving. In the light of the rapidly-growing consumption it is not assuming too much to believe that it will never reach a lower point than it has in the past season. The easily available ores of good quality are, as is well known, very limited in extent and are controlled by companies not desirous of exhausting them at too rapid a rate. The foreign ores of high grade are available chiefly on the coasts. Their transportation far inland would make their cost prohibitive.

By taking all these conditions into consideration, one can feel reasonably certain in calculations of the probable revenue in the development of a low-grade iron-ore proposition, using the present ore-prices in the various furnace-districts as a basis for finding the value of the concentrated and agglomerated product.

It is interesting to compare the various items of cost entering into the production of a high-grade blast-furnace material from a low-grade magnetic ore. We may take as an example an ore averaging 36 per cent. of iron in its crude state and occurring in such a large deposit that for several years to come it can be mined in open quarry and loaded with the steam-shovel. If the intention is to produce concentrates with 64 per cent. of iron, it will be necessary to mine 2 tons of crude ore to make 1 ton of finished product.

The plant is, we will say, laid out for the treatment of 2,000 tons of crude ore daily, which corresponds to 1,000 tons of concentrates in 24 hr. A plant of this tonnage can be considered of normal size. It is assumed that power can be obtained in the form of water-power, and that, transmitted to the works, it does not cost more than $25 per horse-power year.

Labor is calculated at 20 cents per hour. The quarry and coarse-crushing plant are operated only in the day-time because labor enters heavily especially into the quarry-operations, and therefore better economy is obtained by accumulating during the day-shift enough crushed ore, which is distributed in the bins over the fine-crushing plant, to last over night. The quarry- and coarse-crushing machinery is also easily obtainable in large units, making it convenient and economical to handle from 200 to 300 tons per hour.

In such a plant the following items of cost would enter into the cost of producing 1 ton of finished product.

Mining and transportation to crusher	$0.75
Crushing	0.10
Grinding and concentration	0.40
Agglomeration	0.50
Tailings handling and pumps	0.05
Shops, local administration, taxes, insurance	0.15
	$1.95

To this cost of manufacture is to be added the royalty on the ore as mined or as shipped, charges on account of the sinking-fund, selling-expenses, and freight to furnaces. These will probably bring the total estimate for the agglomerated product laid down at the furnace to $3.75 per ton.

As soon as the operations have reached the point where it is necessary to resort to underground mining, the cost of the product will be increased, but the calculations still show a profit in the operations on account of the high quality of the material, with its freedom from phosphorus and sulphur, which will make it bring a premium in the market.

SUBJECT TO REVISION.

[TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS.]

A Graphic Solution of D'Arcy's Formula for the Transmission of Compressed Air in Pipes.

BY NATHANIEL HERZ, LEAD, S. D.

(Cleveland Meeting, October, 1912.)

The formula very frequently used for computing the economical size of pipe to transmit compressed air is that of D'Arcy, as follows:

$$D = c\sqrt{\frac{d^5(p_1 - p_2)}{w_1 l}}$$

Where,

D = the volume of compressed air delivered in cubic feet per minute, at the final pressure.
c = an experimental coefficient determined for the various sizes of pipe.
d = the diameter of the pipe in inches.
l = the length of the pipe in feet.
p_1 = the initial gauge-pressure in pounds per square inch.
p_2 = the final gauge-pressure in pounds per square inch.
w_1 = the density of the air, or the weight in pounds per cubic foot at the initial pressure, p_1.

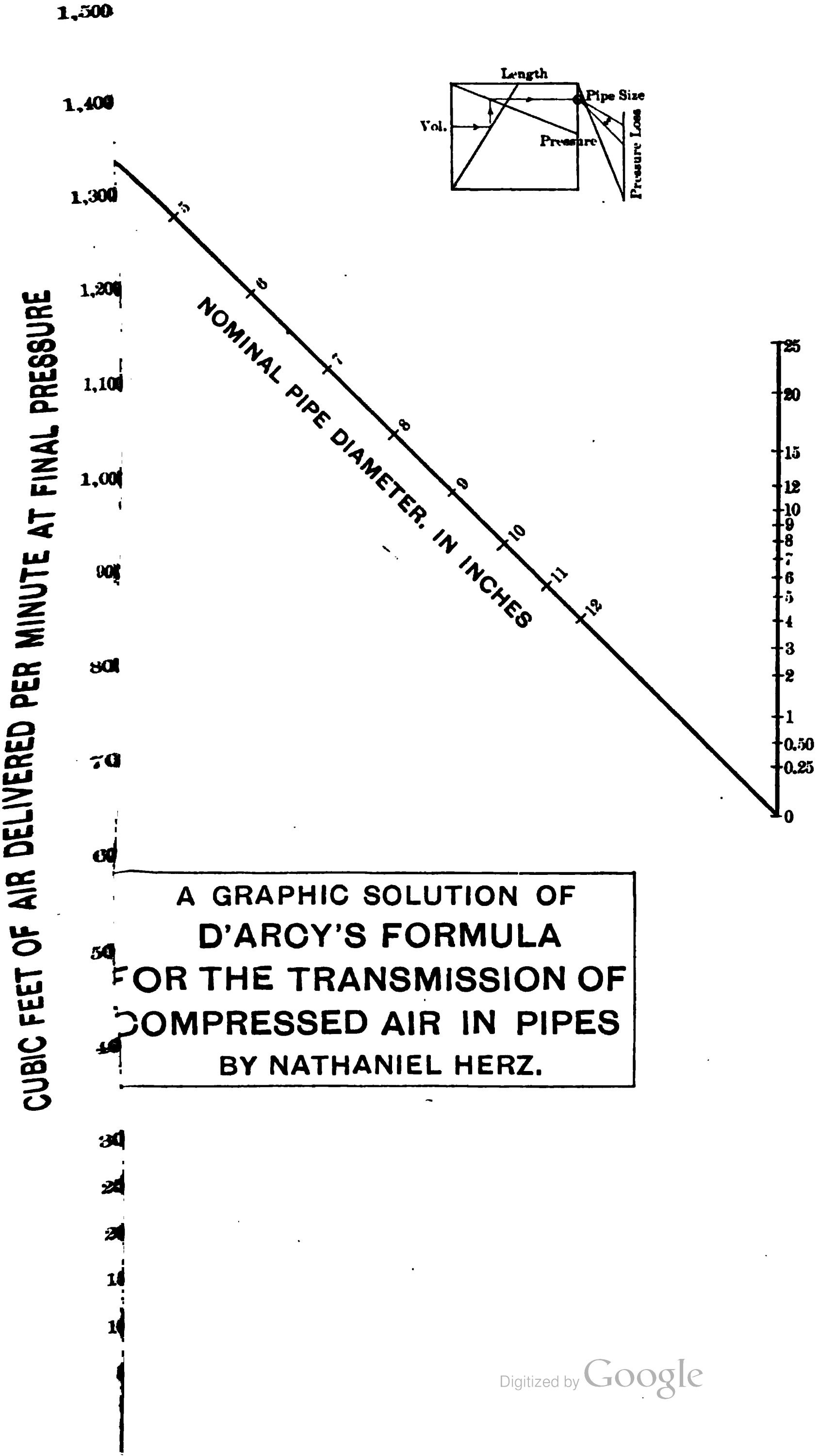
The most common case is that in which the given quantities are: the quantity of air required, the length of the pipe, and the initial pressure. The method of solution is to assume a pressure-loss and to compute the remaining factor, $c\sqrt{d_5}$, thus giving the size of pipe corresponding to the assumed loss of pressure. It is always desirable to try two or more pressure-drops, in order to find the combination that is most satisfactory, since often a small change in the size of pipe will reduce or increase the loss of pressure several pounds. An alternative method is to assume a size of pipe and calculate the corresponding pressure-drop. Each method involves a series of tedious calculations to arrive at the most economical solution, and also requires the use of tables giving the constant, c, the actual diameters corresponding to the nominal pipe-sizes, the density of the air, w_1, and often, for convenience, a table giving the value of the expression $\sqrt{\frac{p_1 - p_2}{w_1}}$.

I have constructed a graphic chart, Fig. 1, for the solution of these problems with no computation, and without the use of tables. The procedure is as follows:

Begin with the quantity of compressed air delivered, on the left-hand vertical scale; follow across horizontally to the intersection with the inclined line corresponding to the length of the pipe-line; pass up vertically to the inclined line corresponding to the initial pressure; then across the chart horizontally to the heavy line at the right of the cross-sectioned part of the chart. The point here found is a pivot-point, which is held with a pencil, pen, or needle-point, and a straight edge placed against it and swung across the "Z" diagram. Any two points on the inclined and vertical lines that are cut by the straight-edge at the same time go together as one solution of the problem, giving a pipe-diameter with its corresponding loss of pressure. By swinging the straight-edge, it is possible to see at a glance how the final pressure is affected by a variation of 1 in. in the pipe-size. Moreover, the size giving the most desirable result is determined at one operation. If the drop is considerable, it may be desirable to adjust the volume to correspond with the new final pressure, and to repeat the operation; but, within ordinary economical limits, the error involved by not doing so is negligible.

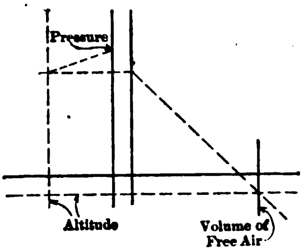
Sometimes the problem may arise in another form—for instance, to find the maximum volume that can be handled in an existing line. In this case, the process is reversed. Begin with the maximum desirable drop, and the size of pipe, then pass to the initial-pressure line in a horizontal direction, then vertically to the length-line, and finally horizontal to the left-hand scale, which will give the corresponding volume. Any other combination can be solved in a similar manner.

The accuracy of this chart is well within commercial limits. It has been checked against calculated values for combinations varying from 100 to 1,000 cu. ft. of compressed air delivered per minute, pressure-losses from 3 to 10 lb., and pipes from 100 to 4,000 ft. long; all results were within 0.5 in. of the pipe-diameter, and most of them within 0.25 in. or less.



A GRAPHIC SOLUTION OF D'ARCY'S FORMULA FOR THE TRANSMISSION OF COMPRESSED AIR IN PIPES

BY NATHANIEL HERZ.



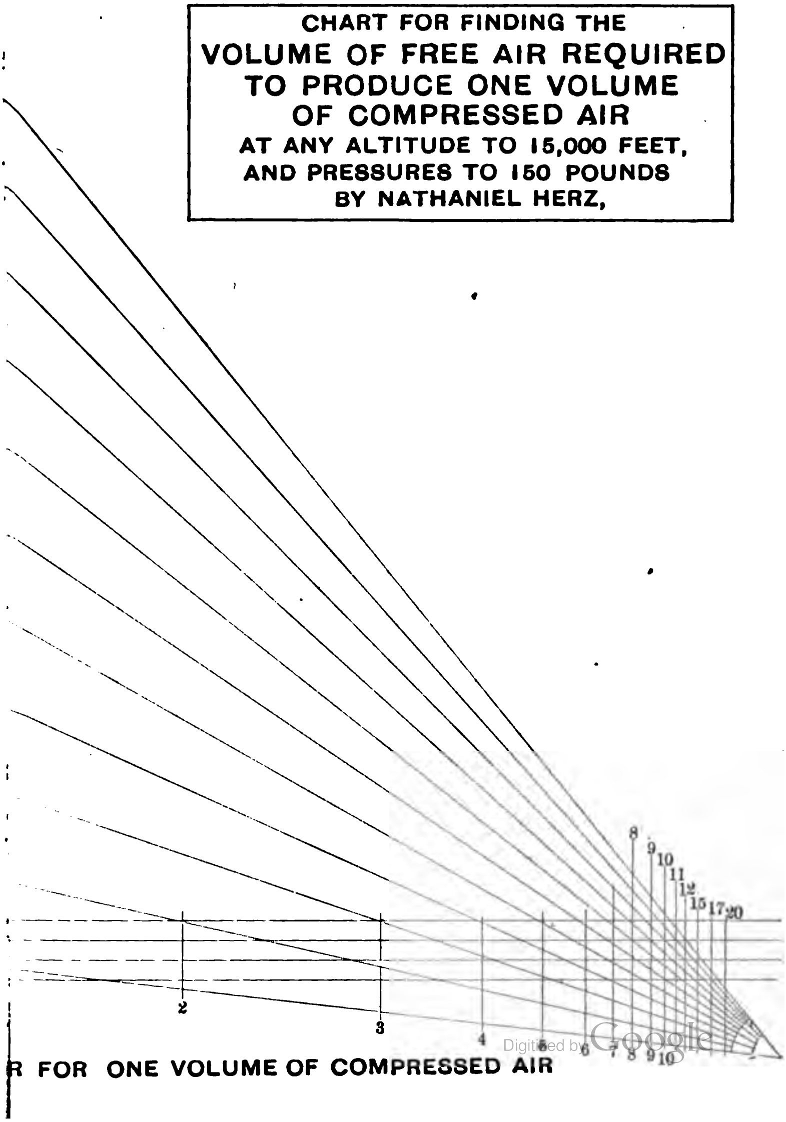


CHART FOR FINDING THE VOLUME OF FREE AIR REQUIRED TO PRODUCE ONE VOLUME OF COMPRESSED AIR

AT ANY ALTITUDE TO 15,000 FEET, AND PRESSURES TO 150 POUNDS

BY NATHANIEL HERZ,

To add to the usefulness of this chart, I have also worked out a straight-line chart, Fig. 2, for finding the relation between the volume of free air at any altitude up to 15,000 ft., and the volume of compressed air at pressures up to 150 lb. per sq. in. This diagram is the pv = c curve with altitude corrections, thrown into a straight-line chart, all the lines intersecting at the lower right-hand corner. It is used as follows:

Begin with the gauge-pressure on the vertical scale of pressures, using the left-hand line; pass parallel to the slightly inclined lines and intersect the vertical line corresponding to the altitude; then go horizontally to the right-hand vertical pressure-scale. Extend a straight-edge from the point just found to the intersection-point in the lower right-hand corner of the chart, and the point where this crosses the horizontal line corresponding to the altitude gives the volume of free air required to produce one volume of compressed air. This chart is most accurate for the lower pressures, but even for higher pressures it is sufficiently close for most purposes that arise in practical work in compressed-air transmission.

SUBJECT TO REVISION.

[TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS.]

The Sulphatizing-Roasting of Copper-Ores and Concentrates.*

BY UTLEY WEDGE, ARDMORE, PA.

(Cleveland Meeting, October, 1912.)

In general, the art of securing copper from sulphide ores or concentrates may be said to consist of: (1) separation, in the molten state, of copper sulphide with some iron sulphide, from the great bulk of gangue-material, followed by (2) elimination of sulphur and iron from the molten matte, by oxidation and fluxing, leaving metallic copper.

The definition is made broad enough to cover the general practice, whether or not the ore or concentrate is given a preliminary roast for the removal of surplus sulphur.

The above process, which I will designate as the smelting process, makes mecessary the presence in the ore, or the addition thereto, of such silica, lime, and iron as may be necessary for the formation of a light and fusible slag, together with sufficient sulphur and iron for the formation of copper-matte, and also the addition to the matte of sufficient silica for the oxidation and fluxing-out of the iron.

These items, together with the cost of fuel sufficient for melting the mass, constitute important items of cost in the copper-smelting process.

Numerous cases where fluxing-materials are scarce or dear, or where fuel is expensive, have caused special attention to be given recently to the wet process of copper-extraction, which may be designated as consisting of forming soluble copper salts which are then washed or leached out of the ore or concentrates by dissolving in water or acid. Metallic copper may then be secured from the solution of copper salts by electrolysis; by cementation; or copper can be brought out of the solution as a hydroxide or other insoluble precipitate and reduced to metallic copper by subsequent melting in a reducing atmosphere.

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* Presented also at the Joint Meeting of the Institute and Section IIIa, Metallurgy and Mining, of the Eighth International Congress of Applied Chemistry, New York, N. Y., September, 1912, and here published by mutual agreement.

The present paper presents facts relating to only one step in the wet process of copper-extraction and deals only with the formation of copper sulphate by roasting copper-ores or concentrates.

One per cent. of copper-content requires, theoretically, 0.5 per cent. of sulphur to form copper sulphate. In practice, it is necessary to have present an excess of sulphur over this theorectical amount. If sufficient sulphur is not present in the material to be treated, it is then necessary to add sulphur in some form, either as sulphur, SO_2, SO_3, or in the form of a sulphide or sulphate or sulphuric acid. Iron sulphides are very desirable in ores, when it is required to sulphatize the copper values, and zinc, lead, lime, and magnesia, if present in forms that will consume sulphur, are undesirable.

The principal reactions which enter into the sulphatizing of copper sulphides are as follows:

CuS decomposes in roasting to $Cu_2S + S$, and the free atom of S burns to SO_2 at temperatures above 350° C. (662° F.). Cu_2S at furnace-temperatures below 500° C. (932° F.) reacts principally as follows;

$$2\,Cu_2S + 5\,O_2 = 2\,CuO + 2\,CuSO_4$$

At higher temperatures, the principal reaction is

$$Cu_2S + 2O_2 = 2\,CuO + S_2$$

Different copper sulphide minerals behave differently at furnace-temperatures, but the above formulas show the tendency common to all copper sulphides to form some quantity of copper oxide at all furnace-temperatures.

In roasting to secure the highest per cent. of water-soluble copper, the constant formation of the copper oxide is the chief obstacle to overcome.

The presence of iron sulphide with copper sulphide in the roasting process assists in sulphatizing the copper oxide thus formed. At furnace-temperatures below 600°C. (1,112°F.) there is some basic iron sulphate formed, which, at temperatures above 530°C. (986°F.), roasts to Fe_2O_3 and SO_3, and the SO_3

gas may combine with the CuO to form $CuSO_4$, which is practically stable at temperatures below 650°C. (1,202°F.), or, the CuO may react directly with the $Fe_2(SO_4)_3$, according to the formula,

$$3\ CuO + Fe_2(SO_4)_3 = Fe_2O_3 + 3\ CuSO_4$$

Even the portion of iron sulphide which roasts to Fe_2O_3 is of some assistance in sulphatizing the CuO, as at a temperature between 500° C. and 750° C. the iron oxide acts as an energetic catalyzing agent for the oxidizing of the SO_2 to SO_3, which in turn tends to combine with the CuO to form $CuSO_4$.

Also, SO_2 generated in the furnace, either from the copper or iron sulphides, tends to sulphatize the copper oxide.

These various principles are generally familiar from enunciation by Aubell, Kothny, Ottokar Hofmann and others, and attention is called to them here to make clear the significance of the following tests made by the Wedge Mechanical Furnace Co., of Philadelphia. The tests were made without the addition of any material to the ore treated:

Test No. 621, *February,* 1910.

Analysis of Ore.

	Per Cent.
Cu	1.07
Fe	49.25
S	32.60
SiO	6.80
MgO	2.18
Al_2O_3	1.59

The copper present was chalcopyrite; iron present was pyrite and pyrrhotite.

Ore was roasted in a Wedge multiple-hearth furnace, with the following results:

	Per Cent.	
Water-soluble copper	0.35	= 27.8 per cent. extraction.
Acid-soluble copper	0.80	= 63.4 per cent. extraction.
Insoluble copper	0.11	
Total	1.26	= 91.2 per cent extraction.

The calcine from the above roast contained 4.7 per cent. of sulphur and was then treated in a Wedge multiple-hearth mechanical muffle-furnace. It was then leached, with the following result:

	Per Cent.	
Water-soluble copper	0.75	= 60.3 per cent. extraction.
Acid-soluble copper	0.48	= 38.1 per cent. extraction.
Insoluble copper	0.02	
Total copper secured in solution		98.4 per cent.

In the above test, the copper designated as acid-soluble copper was copper oxide, soluble in very dilute sulphuric acid.

It will be observed that, in spite of the presence of liberal quantities of iron sulphide, there was considerable formation of copper oxide in the furnace, but this occurred chiefly in the preliminary roast, where, by reason of the considerable excess sulphur, the temperature became excessive. Analysis of the gas leaving the furnace in initial roasting showed formation of sufficient SO_3 for the production of sulphuric acid to recover all of the copper oxide, or, if electrolysis were used in subsequent recovery of the copper from solution, more sulphuric acid than would be required to sulphatize the copper oxide.

Test No. 620, *March*, 1910.

Analysis of Ore.

	Per Cent.
Cu	3.52
Fe	9.76
S	9.54
Insol	66.5

Copper present as chalcopyrite; gangue consisted of schist.

This ore was given a single treatment in a Wedge multiple-hearth mechanical muffle-furnace 9 ft. 9 in. in diameter with five hearths; four hearths muffle-fired with coal.

The analyses made currently during the test showed:

Water-soluble copper, from 60 to 70 per cent. of total copper.
Acid-soluble copper, from 20 to 22 per cent. of total copper.

Total, from 80 to 92 per cent. of total copper.

In this test the SO_2 and SO_3 gases generated in the upper hearths of the furnace were drawn downward over the ore on the lower hearth of the furnace.[1]

If electrolysis were used, the copper sulphate solution would yield, in this case, a great excess of sulphuric acid for the recovery of the copper oxide.

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[1] U. S. Patent No. 966,277, Aug. 2, 1910.

Test No. A–100, *July*, 1911.

Analysis of Ore.

	Per Cent.
Cu	4.38
Fe	9.61
S	16.37
CaO	1.10
MgO	0.72
Insol	61.21

Copper present as chalcopyrite; the gangue was diabase.

This ore was given a single treatment in a Wedge multiple-hearth muffle-furnace[2] having a diameter of 9 ft. 9 in. with five hearths, being the same furnace as was used in the previous test.

Actual leaching of this ore showed a recovery by

Water extraction	73.7 per cent. of copper in ore.
Weak sulphuric acid	15.7 per cent. of copper in ore.
Making a total of	89.4 per cent. of the total copper in ore.

It is believed that this ore, sulphatized in a larger furnace of the Wedge type, would yield still better results.

Tests so far given, were made with ores containing copper as chalcopyrite.

The following tests were made with an ore in which the copper was present chiefly as bornite, with some copper silicate and carbonate.

Sample No. 25, *April*, 1911.

Analysis of Ore.

	Per Cent.
Cu	3.40
Fe	11.05
S	13.02
SiO_2	63.90

This ore was ground to 20-mesh and was given a single treatment in a small furnace test and showed:

Water-soluble copper	80.80 per cent. of total copper-content.
Acid-soluble copper	11.8 per cent. of total copper-content.
Total recovery	92.6 per cent. of total copper-content.

The above results have since been duplicated with this ore in a Wedge mechanical muffle-furnace 20 ft. in diameter with five hearths, and sufficient sulphuric acid is recovered from the SO_3 in gases leaving the furnace to recover the oxide of copper formed and make the extraction as above shown in the subsequent leaching-operation.

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[2] U. S. Patent No. 654,335, July 24, 1900.

Sample No. 7, *December*, 1909.

Analysis of Ore:

	Per Cent.
Cu	1.85
Ni	2.75
Fe	47.78
S	30.85
SiO_2	15.30

It was desired to extract the copper, leaving the nickel in the leached ore, together with the iron.

The analysis of the calcine showed:

Water-soluble	Copper, 69.2 per cent.	Nickel..... 13.6 per cent.
Acid-soluble	Copper, 12.8 per cent.	Nickel..... 4.4 per cent.
Total soluble	Copper, 82.0 per cent.	Nickel..... 18.0 per cent.

The leached ore contained

Copper	0.29 per cent.
Nickel	2.80 per cent.

It has been shown in tests made by the Wedge Mechanical Furnace Co. that copper-ores containing little or no sulphur or iron can be treated by sulphatizing-roasting by the addition of iron sulphide in the form of iron pyrites.

Test No. 604, *August*, 1909.

Analysis of Ore.

	Per Cent.
Cu	3.14
Fe	2.25
S	none
Insoluble	85.5

This ore contained copper as carbonate and the gangue was porphyry.

This ore was mixed with 5 per cent. of iron pyrites containing 47.5 per cent. of sulphur.

The test was made in a 9 ft. 9 in. Wedge mechanical muffle-furnace with five hearths, and the analysis of the calcine showed 92 per cent. of the copper to be soluble in water. The copper solution was very pure, containing only from 0.1 to 0.2 per cent. of Fe_2O_3 to 7 per cent. of copper sulphate. Copper precipitated from the solution with clean scrap-iron tested 96 per cent. pure cement copper.

Test No. 634, *November*, 1910.

Analysis of Ore.

	Per Cent.
Cu	1.86
Fe	2.79
S	0.80
Al_2O_3	2.58
Insoluble	88.3
Combined water	3.4

The sulphur contained in this ore seemed to be partly in cinders from the locomotive, which accumulated in the open car during transit from the West. Copper was present largely as copper silicate. Six per cent. of pyrites containing 47 per cent. of sulphur was added to this ore and this mixture was given a single treatment in a Wedge mechanical furnace, with five hearths and a preheating hearth, muffle-fired with coal; diameter of furnace 9 ft. 9 in. Tests were made of the material as it progressed through the furnace from hearth No. 1 (top hearth) and showed as follows:

Leaving Hearth No. 1	79.3 per cent. of copper water-soluble.
Leaving Hearth No. 2	81.4 per cent. of copper water-soluble.
Leaving Hearth No. 3	93.4 per cent. of copper water-soluble.
Leaving Hearth No. 4	95.2 per cent. of copper water-soluble.
Leaving Hearth No. 5	95.2 per cent. of copper water-soluble.

There was no heat applied to the fifth hearth and the sulphatizing was completed in the fourth hearth. This calcine leached readily with cold water, using 1 ton of water per ton of ore. Solution secured was used on successive batches and brought up to 15 per cent. of $CuSO_4$ containing only 0.1 per cent. of Fe. The tailings from this leaching-test contained Cu 0.08 per cent.

Up to this time, very few records are available from furnaces of large size. The furnace with which most of these records were made was a furnace 9 ft. 9 in. in diameter and 12 ft. high, with five hearths and a preheating hearth. This furnace, a generation ago, would have been considered a large furnace, even with a capacity of only one carload of ore per week, but the commercial problems of to-day call for much larger appliances. Several large mechanical muffle-furnaces are now being installed by large copper companies, and records from furnaces with an output of from 20 to 40 tons daily, per furnace, will soon be available.

The above is a record indicating in advance the possibilities of a process which may soon, in special cases, displace the smelting process of copper-production.

SUBJECT TO REVISION.

[TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS.]

The Development of the Parkes Process in the United States.*

BY ERNST F. EURICH, NEW YORK, N. Y.

(Cleveland Meeting, October, 1912.)

ALEXANDER PARKES patented in England in 1851–52–53 a process for desilvering lead by means of zinc, making use of the greater affinity of silver for zinc than for lead, discovered by Karsten in 1842. The efforts in England to develop it commercially were not particularly successful, nor was the attempted introduction into the United States. This last was not surprising, as it was about the beginning of our Civil War, and at that period there was very little lead-bullion produced in the United States and there was no field for the process here. There was at that time on Staten Island an establishment that pattisonized lead, probably from Rossie, N. Y., and Chester, Pa., and possibly some Spanish lead, but the production of lead-bullion in the Far West had not yet begun. In 1864, Edward Balbach, of Newark, N. J., patented in this country a process for desilvering lead by means of zinc, and in 1867 Edward Balbach, Jr., patented a "movable black-lead retort with a neck, placed in a furnace," for the purpose of distilling the alloy of zinc, lead, and the precious metals.

A brief comparison of the Parkes and the Balbach processes is interesting. The Parkes process as practiced in England consisted in stirring into silver-bearing lead, melted in a kettle, from 1 to 2 per cent. of zinc, cooling, skimming the alloy of lead, zinc, and silver that rose to the surface, liquating it in an iron retort, distilling the residual rich alloy in fire-clay retorts, thus regaining the zinc, and refining the desilvered lead at a low heat in a shallow cast-iron pan that formed the hearth of a reverberatory furnace. If the lead was too impure, it was first softened in a reverberatory furnace similar to the refining-furnace. The Balbach process liquated the softened bullion in a reverberatory with a cast-iron inclined hearth, the eliquated metal flowing into a kettle which contained the requisite quantity of melted zinc, thoroughly stirred the contents and ladled into molds, transferred the bars to a second liquating-furnace, where they were heated to the melting-point of lead. The eliquated lead was to be measurably free from silver, the residual rich alloy was distilled in a black-lead retort, the zinc given off condensed and the rich lead resulting cupelled. The desilvered lead was refined at a high heat in a reverberatory built of fire-brick. Impure lead-bullion was first softened in a reverberatory furnace similar to the one used for refining. The Balbach process as patented had some of the features of the Parkes, and after several years of operation, about 1872, was so modified as to resemble it more closely.

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* Presented also at the Joint Meeting of the Institute and Section IIIa, Metallurgy and Mining, of the Eighth International Congress of Applied Chemistry, New York, September, 1912, and here published by mutual agreement.

It is interesting to note that in the desilvering of lead-bullion by means of zinc as practiced to-day, the different operations are fundamentally identical with those outlined by Parkes. The failure of the process, or rather the lack of commercial success, was due solely to the apparatus employed; the shallow cast-iron pans frequently cracked, and the fire-clay retorts were not adapted to the work required of them. By the use of fire-brick furnaces for softening and refining, and of black-lead retorts for distilling, those works which later, in the United States, followed the lead of Parkes in their operations, made the Parkes process an unqualified success. This is the process by which most of the lead-bullion in the civilized world is desilvered, and in many respects it has found its best developments in American plants.

With the opening of the Pacific railroads, the exploitation of the Far West received a great impetus. Furnaces for smelting lead- and silver-ores were soon in operation, and the lead-bullion they produced, as well as some of the ore mined, sought a market in the East. Smelting- and refining-works were built at different points from the Atlantic to the Pacific, and by 1873 there were 14 establishments in operation at which smelting of ores and refining of lead-bullion was carried on. Two of these, the Pennsylvania Lead Co., near Pittsburg, Pa., and the St. Louis Smelting & Refining Co., near St. Louis, Mo., certainly the former, followed exactly the lines laid down by Parkes, using, however, black-lead retorts and fire-brick softening- and refining-furnaces. The Germania works at Salt Lake City soon followed suit, and of the remaining refineries, the Balbach works at Newark, N. J., the Omaha works at Omaha, Neb., and the Selby works at San Francisco were soon the only survivors. These three used the Balbach process, but it had been so modified as to approach closely to the Parkes model. By the end of the 70's or the early 80's various processes had been tried out, and only the Parkes remained.

However much the American works differed in the details of operations, there was one feature that soon became common to all, the terrace construction of the desilvering-plant. The softening-furnace occupied the highest level, the desilvering-kettle the next, the refining-furnace below this, and lowest of all was the floor on which were the molds for the merchant lead. The charge finished in one apparatus was tapped and flowed into the next, and from the last was run into the molds.

To trace the improvements in the construction and operation of the constituents of the plant, it will be simplest to consider each operation separately.

Softening Lead-Bullion.—This operation is carried out in a reverberatory furnace. We early find the hearth of the softening-furnace built inside a riveted sheet-iron pan, the material of the hearth being fire-brick. In this construction, in use in 1873 and possibly earlier, the wrought-iron prevents all leakage of lead out of the furnace, and the bottom, of fire-brick, laid as an inverted arch of from 6 to 9 in. rise, effectually resists the plumbostatic pressure of the charge. Adopted later in Europe and wherever the Parkes process is used, it has become the standard construction.

To counteract the corroding action of the oxides at the lead-line, and reduce the frequency of repairs, water-cooling in various forms was resorted to. A 2.5- or 3-in. water-pipe was placed along the lead-line next to the pan, or the sides and ends of the hearth were provided with water-jackets, or again the pan proper was placed in a second riveted pan, leaving a space between the sides and bottom in which water circulated. This last was eventually abandoned, as the cooling of the bottom was not required and increased the fuel-consumption. With the advent of magnesite bricks, these were used along the lead-line in addition to the water-cooling. Indeed, in some cases they have led to the abandoning of water-cooling except under the skimming-door.

With the increase of the regular supply of lead-bullion that could be depended on, the size of the softening-furnace increased from a capacity of 10 tons to 25, 40 and up to 90 tons. With hand-charging this was about the economical limit, but the introduction of a charging-apparatus (first used at the Natonal works of the American Smelting & Refining Co.), which by means of an endless belt elevated the pigs and dropped them into a chute down which they slid into the furnace through a hole in the roof, has made still larger furnaces feasible, and a from 210- to 240-ton furnace is now in operation at Omaha. In the matter of size of furnaces, England was for quite a time in the lead, as 140-ton furnaces were in use there for a number of years before furnaces of a similar size were built here. The time required for refining a large charge in a properly proportioned furnace is no longer than is required for a smaller one, and, provided sufficient lead-bullion is available to keep it in steady operation, a large furnace is more economical in labor, fuel, and repairs than a smaller one.

The proper method of conducting softening-operations has long been known, but not always properly observed. When its laws were transgressed the penalty followed,—a smaller direct yield of soft lead, a larger quantity of secondary products to be reworked, and greater costs. It is not necessary to go minutely into the details of the operations. The important points are: melting the charge at a low heat and removing the dross that accumulates on the surface, which contains the greater part of the copper present; next, increasing the heat, admitting air, oxidizing the impurities susceptible of oxidation (mainly arsenic and antimony) and skimming off the arsenical and antimonial lead oxides. In some works crushed litharge free from copper was regularly used for the purpose of oxidizing the impurities. The advantages of reducing the lead of the litharge and at the same time oxidizing the impurities are evident. It is quite noticeable that as soon as the arsenic is pretty well eliminated, the oxidation of the charge proceeds more rapidly, and it is for oxidizing the arsenic that litharge has been found particularly helpful.

For a time the lead-smelter in the West, with few exceptions, loaded into the lead-bullion which he shipped, as much as possible of the dross the furnace made. Gradually it was appreciated that this procedure was not to his advantage, and it was modified. The Philadelphia Smelting & Refining Co., at Pueblo, Colo., was the first systematically to prepare clean bullion for shipment. It was drossed in the cooling-pot before ladling, the resulting pigs were remelted in a large kettle holding a carload and, after careful drossing, run into molds. At the furnaces controlled by the American Smelting & Refining Co. the same practice is now followed. Such preliminary drossing, carefully carried out, renders drossing in the softening-furnace unnecessary, and softening then consists merely in eliminating the arsenic and antimony. After softening, the charge is tapped into a launder, through which it flows into the desilvering-kettle.

Desilvering With Zinc.—The kettles employed for this operation originally had a capacity of 10 tons, but this capacity has been increased to 60 or 70 tons. At first kettles were cast with a spout at the bottom through which the charge was tapped into the refining-furnace. In 1877, at the works of the St. Louis Smelting & Refining Co., the use of a siphon for transferring the charge from the desilvering-kettle was introduced by A. Steitz. His example was followed by other works and the spout, no longer necessary, disappeared. In recent years a horizontal centrifugal pump driven by an electric motor has come into use for emptying the kettles. With large kettles, the centrifugal pump saves time. Furthermore, its use will affect the design of the kettles and will have an influence in the arrangement of the constituent parts of plants that may be erected in the future, and may use it. No longer limited to a depth of 40 in., the kettle can be made deeper and capacities in excess of 60 tons will be entirely feasible.

The apparatus for liquating silver crusts varied at different works. At some it was a reverberatory furnace with an inclined cast-iron hearth; at others, a kettle with or without a spout. In the former case an additional kettle was provided to receive the eliquated lead. These various devices have been superseded by the silver-crust press, which, producing a richer crust, is now in general use. It was first introduced at the works of the Pueblo Smelting & Refining Co. by W. H. Howard, its inventor and patentee.

It is of prime importance that the zinc which is to effect desilvering be well incorporated with the lead-bullion. At first this was done solely by manual labor. In the later 70's steam-stirring was introduced at the works of the Pennsylvania Lead Co. It consisted in passing dry steam through the charge, producing a circulation which, combined with a rabbling of the surface, effected a good incorporation of the zinc which had been previously melted on the surface. Both of these methods have given way to the mechanical stirrer, invented by W. H. Howard and patented in 1894 and first used by the Pueblo Smelting & Refining Co. Like its prototype, the device of Cordurié, it uses a propeller-wheel for establishing a circulation and incorporating the zinc. The details of its construction, however, make it more efficient. It works by steam or electrical power and effects a thorough and uniform mixing of the metals.

For quite a time there was no material change in the manner of conducting zincings from that practiced in the early trials of the Parkes process in England. The general custom was to add the zinc in three doses, stirring in each addition, and then cooling and skimming until a ring of chilled lead had formed around the kettle. Only the first crust was liquated and the dry crust sent to the retorts; the second and third crusts were used again in the next charge. Some works used four zincings, the first to collect the gold along with a small part of the silver in a so-called gold crust. When this was done, the first and second crusts were liquated separately and the dry crusts were worked up each for itself.

With a wide range in the silver-content of the lead-bullion received from various sources, it was desirable to have tables for the zinc additions. Lead desilvered by zinc contains at its melting-point 0.6 per cent. of zinc. The total quantity of zinc required to extract the silver from any given quantity of lead-bullion is represented by a formula containing a constant (0.6 per cent. of the weight of the charge) plus a variable that increases with the silver-content, but at a lower ratio. On this basis, tables for zinc additions were worked out covering a wide range of silver-content.

The time required for desilvering a charge was from 15 to 20 hr. Attempts to reduce the time by reducing the number of zincings were successful, but at the expense of larger zinc additions. That such was the case was due to the incomplete incorporation of the zinc. With the use of the Howard stirrer, supplemented by the Howard press, the total time of desilvering has been reduced to not to exceed 10 hr. The zinc additions are not greater and there are no longer coolings to a "ring," until the very last, when the silver-content is to be brought down to the assay required in the merchant lead.

Refining.—Eliminating the zinc retained by the desilvered lead has, in the United States, been effected either in a reverberatory by oxidation at a comparatively high heat, or in a kettle, by passing dry steam through the lead previously heated to redness, as was first practiced by Cordurié. Steaming in kettles was used by some of the early unsuccessful works; it has been repeatedly tried experimentally, the Pueblo Smelting & Refining Co. alone adopting it permanently for refining. The use of the reverberatory has found most favor with American refiners, and at the present time it is exclusively employed.

The construction of the refining-furnace is exactly the same as that of the one used in softening. Its size has kept pace with that of the softener, so that while the charge of a large softener is divided among two or more desilvering-kettles, the contents of these kettles are again gathered together for treatment in one large refiner.

After the zincky oxides have been skimmed, the ordinary practice is to raise the heat for a short time and to admit air, in order to remove the last of the antimony. At some works this final removal of the antimony is accomplished by steaming for a short time without raising the heat. The same procedure has been employed in accelerating the elimination of the antimony in the softening-operations.

Tapping the refined lead into a merchant kettle and ladling into molds were early replaced by more expeditious methods. In 1876, at the Pennsylvania Lead Co.'s works, this was done by running the lead through a swinging 1.5-in. pipe into molds arranged in the segment of a circle, the flow being controlled by a cast-iron cock. With the introduction of the siphon, some works continued to use the merchant kettle and employed the siphon to run the lead into molds.

The next improvement was the molding-machine, a moving endless belt on which the molds are mounted. As each mold comes under the discharge-pipe it is filled, is then carried forward, cooled, and as it passes over the end sprocket-wheel the pig of lead is dumped. This machine, introduced at the National works by the then superintendent, A. Oechsler, is especially adapted for handling large charges economically.

Distilling the Zinc Crust.—The early use of the black-lead retort by Balbach has already been noted. It was mounted in an inclined position in a coke crucible-furnace, the neck of the retort projecting through the front wall, the zinc distilled off being condensed. When the charge was finished, the supporting front wall was removed, the crucible seized by tongs, turned down and emptied. A. Faber du Faur, in 1870, swung the crucible-furnace on trunnions, so that by tilting it the retort could be emptied. This tilting furnace is used to-day by all the lead-refiners in the United States, and has been largely adopted in Europe.

About 1865 the Parkes process was taken up on the continent of Europe. It is interesting to note the various ways that were followed in treating the zinc crust. At some works it was smelted in a blast-furnace and the resulting lead cupelled; at others it was scorified on a lead bath. It was also treated with steam to oxidize the zinc, the resulting oxides being scorified on a test or treated with acids. In all cases the zinc was sacrificed and there was an appreciable silver-loss. These various methods are in striking contrast to the simplicity of the original Parkes method made practicable by the American modifications.

From a capacity of 250 lb. the retorts were gradually increased in size until a capacity of from 1,200 to 1,400 lb. had been reached. As the tilting furnace was a patented device, it was not used by all refiners during the life of the patent. Various furnace constructions were devised, some using coke, others coal, and all using the same kind of retort, but instead of emptying the retort by tilting, the contents were tapped or dipped out. With the expiration of the patent, these various constructions were replaced by the Faber du Faur tilting furnace. The only change in the operation has been the substitution of fuel-oil for coke. The advantage is not so much in reduced fuel-costs, but rather in the longer life of the retort, the more uniform and better results obtained, and in the fact that the work is easier for the men.

Cupelling Rich Lead.—Although cupelling rich lead is not peculiar to the Parkes process, and, together with the treatment of the various secondary products, belongs rather to the general metallurgy of lead, yet as the large tonnage handled by American works using the Parkes process has brought about a characteristic development of this operation, a brief sketch of its development in this country will not be out of place.

In the United States, cupellation has always been effected in the English type of cupel-furnace. Small at first, they have increased gradually from a test-capacity of 2,000 lb. or less to one of 6 tons. The use of bone-ash for making the test soon gave way to other, cheaper and more satisfactory, materials. Varying with individual preference, there have been used Portland cement, mixtures of Portland cement with ground fire-brick, Portland cement with ground limestone, ground limestone with fire-clay; and test frames have been lined with magnesite brick. At present, a mixture of crushed or ground limestone (preferably dolomite) with fire-clay is generally used. To counteract the corrosion of the test by litharge, and the eating out of the litharge channels, various water-jacket devices have been introduced; A. Steitz, at the St. Louis works, being the first to use water-cooling for this purpose.

The test of the English cupelling-furnace, while it is removable, is stationary when put in position in the furnace. American practice early developed a tilting test, and both stationary and tilting tests are at present in use. The latter is mounted in such a manner that, by means of a differential block or by a hand-wheel, it can be tilted up or down and the flow of the litharge controlled. The prototype of this device is found in a silver-refining furnace in use in Sweden at Konigsberg in 1854. The tilting test was especially convenient before the introduction of the litharge water-jackets, and even under present conditions is preferable to the stationary test.

For a long time it was customary to cast the rich lead from the retorts into bars and gradually feed these bars into the cupelling-furnace. A marked improvement was effected about 1901, when F. B. F. Rhodes, at the National works at South Chicago, poured the charge from the retort into a pot, raised the pot by means of a pneumatic lift and poured the contents into the test. This procedure has been followed with advantage at other works.

Thus the simple small English cupelling-furnace has been developed into its American descendant, which, in its most complete form, has a tilting, water-cooled test holding 6 tons or more of rich lead, is capable of oxidizing 7 tons of lead in 24 hr., and produces doré silver 998 fine. Depending on the method of parting to be used, whether by electrolysis or sulphuric acid, the contents are either ladled into thin slabs, or they are poured into molds by tilting the test.

Except in the case of the retorts, no reference has been made to the use of oil as fuel. In a general way it will suffice to say that where the price warrants, as at the Selby works at San Francisco, Cal., it has satisfactorily taken the place of coal.

SUBJECT TO REVISION.

[TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS.]

The Effect of Alumina in Blast-Furnace Slags.

Discussion of the paper of Mr. J. E. Johnson, Jr., presented at the Cleveland meeting October, 1912, and printed in *Bulletin No.* 70, October, 1912, pp. 1123 to 1132.

R. H. SWEETSER, Columbus, Ohio:—I think that Mr. Johnson has opened up one of the most profitable discussions that could come to blast-furnace men, and I am very glad that he has come out so strongly against the old oxygen ratios. I will confess I never could understand them myself and never used them in practice. Now, I disagreed with Mr. Johnson strongly three years ago, when he first showed me his paper, and I thought that the question could be approached from the silica-alumina ratio side rather than from the lime-silica side. Eight years ago at "the Soo" we had an ore that absolutely refused to "behave in the furnace," and we could not use it; but going on the theory that all ores can be used in a blast-furnace if it is known how to handle them, we went over our records very carefully to find out when we could use ores of that nature successfully, and when we could not; and we found that the ore that was giving a great deal of trouble had a silica-alumina ratio that was very, very high—in fact, it had practically no alumina at all, and our slags had a very small percentage of alumina. I had experience in Tennessee with the Cranberry ore, which gave alumina in the slag down as low as 4.5 per cent., and I could not understand why the ore we had at "the Soo" could not be used as successfully. I located on a triaxial diagram the analyses of good and bad slags, and it seemed that the bad slags had a ratio of from five to eight of silica to one of alumina. Then I concluded that the silica-alumina ratio in our slag was too high, and that we could correct our trouble with this particular Mesabi ore by lowering the silica-alumina ratio. We made this change, and used the ore successfully. I have since learned that the "silica-alumina ratio" was partly right, and that Mr. Johnson's theory was partly right, for practical purposes; but neither explanation gives the true solution of the function of alumina in blast-furnace slags.

President Kemp spoke about a thing to-day that comes I believe nearer to the practical solution, and that is that the blast-furnace slags are mixtures of minerals, and that the working of slag depends on the melting-point of all the minerals that go to make up that slag.

I am indebted for a lot of information on this subject to Wilbur Stout, now with the Ohio State Geological Survey, who said that slags were really combinations of minerals, and that the melting-point would vary according to the number of minerals that were present, and that the melting-point would be governed greatly by the predominance of certain minerals. Mr. Frantz, who was mentioned in Mr. Johnson's paper, and I have worked over this a good deal together, and last week Mr. Frantz showed me a line across one of these triaxial diagram charts that seemed to explain the reason why alumina acts sometimes as a base and sometimes as an acid. We have actually replaced lime in our slags with alumina, and it seemed there was no change in the character of the slag; you could not tell, from its appearance, when there was 20 or only 14 per cent. of alumina present—the slags would look about the same to the blower, yet those slags varied greatly in the composition. I think there is a big field here if some of the scientists will take up this matter of slag analysis and determine these melting-points. The charts that have already been published have been prepared by men working in geology or in cement, but nothing has been done along the line of actual blast-furnace slags. I think it so necessary to explore the field that some one ought to take it up at once.

C. H. Fulton, Cleveland, Ohio:—A Scandinavian geologist, J. H. L. Vogt, has done a great deal of work on the mineral constitution of slags, and has correlated this work with the newer ideas of physical chemistry, constructing equilibrium diagrams such as are used for alloys, one of which was shown by Mr. Stoughton this morning. When the slags are looked upon from this view-point, we find that the entities that enter into a slag composition are minerals, and fortunately, most slags are composed of two minerals only. On this basis the action of alumina is rather simply explained, if we do not go too far into the theory of the subject. For instance, alumina may play the part of both an acid and a base. Whether it plays one or the other or both parts, depends entirely upon the mineral composition of the slag as a whole. If the chemical composition of the slag is such that the minerals augite, anorthite, melilite or gehlenite may form, alumina acts as a base. If alumina is present in greater quantity than can be carried by the amount of these minerals formed, the excess performs the part of an acid. We find in certain slag compositions which contain alumina, that this will form spinel under the influence of the magnesia present. The practical point involved in the formation of spinel is this: the spinel (that mineral in which the alumina is the acid), has an extremely high melting-point, and it freezes out from the slag at such a high temperature that these slags are practically impossible in the furnace, especially when the spinel is present in large proportion. Widely varying percentages of alumina may be carried in slags, dependent upon what the rest of the base and silica contents are. What must be looked out for is that certain types of minerals are formed. If we know what these types are, and what their composition is, we have the key to the slag question. It is very difficult without charts to show the relation of mineral composition to chemical composition, but we find that for iron blast-furnace slags, in order to carry large amounts of alumina the silicate degree must be in the neighborhood of 1.00, with magnesia very low, otherwise such minerals as melilite and anorthite will not form, with the result that part of the alumina goes into the spinel form. There has been more work done on this subject than is indicated by what has been said to-night. It is mostly foreign work of scientific character. I had occasion recently to investigate the reverberatory furnace slags of a Western smelting company, to determine the influence of composition on melting-point, and the relation between melting-point and fluidity-point, or "free-flowing temperature," as Mr. Johnson calls it. It is encouraging to note the results which may be obtained for slags by an investigation based on the equilibrium diagram method such as was before the Institute this morning for iron-carbon.

MR. JOHNSON: I desire to say that I did not mean to belittle the amount of work done; all that I meant to say was that it had all been useless. The scientific people have determined the softening-temperatures of these slags, but they do not interest us. The slag has to be run out of the furnace, primarily, and able to sulphurize the iron secondarily, or it is no good. What is needed is to know the temperatures at which these slags will run. We need an absolute viscosity test, to know that a slag of a certain viscosity will handle itself properly in the furnace at a certain temperature, and until we get results in that shape we cannot use them.

James F. Kemp, New York, N. Y.:—Some very interesting work in this connection has been done in the Geophysical Laboratory of the Carnegie Institution of Washington. Arthur L. Day and E. T. Allen, of the staff, have been giving special study to the melting-points of the feldspars and other minerals.[1] The difficulty of determining the fusing-point in earlier years arose from the uncertainty attached to direct observations with the eye. A change of state from solid to liquid in a brightly glowing mass which sometimes passes through a viscous stage is not easy of accurate definition. Platinum weights were tried by previous investigators. When these sank, fusion was assumed. The later plan is to follow temperature changes with a thermo-couple. When the regularly rising temperature is suddenly accelerated by the latent heat, released by the change of state from solid to liquid, fusion is obviously taking place. Accurate and consistent records are thus obtained.

I was reminded, when Mr. Sweetser was speaking, of the work of Prof. J. H. L. Vogt, formerly of Christiania, Norway, now of Trondhjem. Some hundreds of analyses of slags were compiled by Professor Vogt, and the minerals which had crystallized from them when such developed were also studied. Usually in the case of slag the fused mass chills so quickly that seldom do we get crystallized minerals. But occasionally, with a slow chill, a cavity forms in the midst of the pot, into which well-developed crystals project. By slow cooling also an interlaced mass of minerals like an igneous rock may develop; or a commingled mass of minerals and glass. As Professor Fulton has said, by comparing the known analysis of the slag with the results in crystallized minerals, Professor Vogt has brought out some very interesting relationships. The conclusions have also been of much importance in geological work because of the light cast upon the problems of the lavas and other igneous rocks. The study of Professor Vogt's paper I am sure would be of interest to many of our members engaged in iron smelting, since they could compare the composition of slags with their crystallized results and get some very interesting light. Professor Vogt's paper is published in German, and therefore is quite easy to read, if one has that language at command.[2]

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[1] A. L. Day and E. T. Allen, The Isomorphism and Thermal Properties of the Feldspars. Carnegie Institution, 1905. Summary in the *American Journal of Science*, Feb., 1905, p. 93. Contributions on other mineral groups have since appeared from the Geophysical Laboratory.

JOHN J. PORTER, Staunton, Va. (communication to the Secretary *):—Mr. Johnson's paper is particularly timely and interesting because it deals with a phase of operating efficiency. The problem of high cost of living applies to manufacturing-plants as well as individuals, and its best solution is to be found in closer study of all the details of operation rather than in hasty and often haphazard expenditures for plant-equipment. If the best possible were always done with the equipment at hand, it would in many cases be found unnecessary to spend large additional sums, and in other cases would enable the money to be spent to far better advantage.

The subject of blast-furnace slags is a large one, and several points deserving discussion occur to me. I shall, however, confine myself to the one factor of formation-temperature which has been overlooked or ignored by Mr. Johnson, although it has, in my estimation, a decided bearing on fuel consumption.

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* Received Nov. 9, 1912.

[2] Professor Vogt has published a long series of papers beginning as far back as 1884. The best one for the purpose above suggested is Beiträge zur Kenntniss der Gesetze der Mineralbildung in Schmelzmassen und in den neovulkanischen Ergussgesteinen. (Contributions to our knowledge of the laws of mineral formation in smelting products and in the later volcanic rocks. Christiania, Alb. Cammermeyer, Publisher, 1892. Pp. 271, pl. I., II.). This paper has been since followed by two other monographs, Die Silikatschmelzlösungen: Part I., Ueber die Mineralbildung in Silikatschmelzlösungen, 1903; Part II., Ueber die Schmelzpunkt-Erniedrigung der Silikatschmelzlösungen, 1904. (Fused Solutions of Silicates: Part I., On the Formation of Minerals in Fused Solutions of Silicates; Part II., On the Lowering of the Fusing-Point in Fused Solutions of Silicates.) Both these papers were originally published in the Memoirs of the Academy of Sciences in Christiania, but were separately issued by the publishing house of Jacob Dybwad. Professor Vogt has continued to write in later years upon the questions of eutectics and related themes as applied to igneous rocks, but with less immediate reference to slags.

My attention was first called to this point while studying conditions at certain Southern furnaces which experienced much trouble with viscid slag, and which were forced to use an excess of coke in order to correct this difficulty. The trouble had been attributed to high alumina on the basis of the theory discussed by Mr. Johnson, and this was apparently confirmed by the fact that substitution of other ores lower in alumina relieved the trouble. However, the range of alumina in the slag was only from 12 to 15 per cent. or about what is considered good practice with Lake ores, and a neighboring furnace working on a different ore-mixture, but making a slag of practically identical composition, experienced no trouble whatever.

The clue to the problem was given by some observations of slag-temperature taken by a Wanner optical pyrometer. It was found that at one of the furnaces in question, while the slag was excessively viscid, the temperature averaged over 100° F. lower than at another furnace working under similar conditions but using other ores not causing viscid slag. When excessive fuel was used to prevent viscosity of the slag the temperature became only normal, and there was no evidence that the slag was excessively superheated as one would naturally expect. The evidence, therefore, indicated that the viscosity was due to low temperature, rather than to high alumina or other defect in composition.

The difficulty was obviously associated with the use of high percentages of certain "hard red" or Clinton ores of the following approximate composition: Fe, 37; SiO_2, 11; Al_2O_3, 6; and CaO, 11 per cent. Inspection showed that the silica was intimately associated mechanically with the lime and was not, as is the case with many of the Clinton ores, in the form of quartz grains. The substitution of either brown or "soft red" ore up to the extent of 50 per cent. of the burden caused the work of the furnace to become normal.

The theory which I evolved to account for these phenomena was as follows:

Slag is obviously formed in the molten state under the conditions prevailing in the blast-furnace, for it is only by the liquefaction or softening, and mutual solution of the constituents that they can come into the intimate contact necessary for chemical combination. After the slag has melted, however, it flows very quickly into the space below the tuyeres, and out of the zone of maximum temperature. It is generally admitted that slag can take up very little heat after melting, as it drops too quickly into the relatively cool space below the tuyeres.[3] Hence the temperature of the slag will be controlled very largely by its temperature of formation, and if this is low the slag may be insufficiently superheated above the melting-point, and hence viscid.

Now the temperature of formation is dependent very largely on the intimacy of contact of the slag-forming ingredients. If the gangue of the ore is chiefly in the form of grains of sand or layers of slate and the lime is added as lumps of limestone the points of contact will be few, the formation of slag will take place slowly, and temperature of formation will be high because the constituents will descend far down in the bosh to a zone of high temperature before combination is effected. Moreover the slag when formed will be highly superheated, hence fluid, and the furnace will work smoothly, and have little tendency to hang in the boshes.

On the other hand, if the ingredients be in very intimate contact the formation-temperature will be much lower, possibly even below the melting-point, as in portland cement manufacture. In the case described the gangue of the ore contains: SiO_2, 39.3; Al_2O_3, 21.4; CaO, 39.3; total, 100 per cent. Obviously this is a very fusible mixture and will, therefore, melt high up in the boshes and trickling down, will dissolve the lumps of lime and the coke ash before these have had a chance to become highly superheated. The slag not having reached a high temperature before melting will not be sufficiently superheated to be thoroughly fluid, and incidentally may cause more or less hanging on the bosh wall. Its temperature can, of course, be increased by increasing the fuel, but it is obvious that it will take more fuel to raise the temperature 100° when the molten slag is only in contact with the hot gases for a few moments, as in the last case, than when the solid slag constituents remain in the heated zone for a much longer time, as in the first case.

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[3] Of course there will be some superheating after melting, and this will be greater if the slag is viscid, as explained by Mr. Johnson. His idea of the utility of alumina in making high-silicon iron, is in line with what little information I have on the subject.

Undoubtedly it is possible to go too far in the matter of lessening the points of contact of slag constituents, and I certainly would not advocate increasing the size of the limestone, for example, or abandoning entirely the use of self-fluxing ores. On the contrary I believe that in only a few of the Clinton ores are the gangue constituents in such a chemical and mechanical relation as to produce this trouble. Where the silica is present either as grains of sand or as layers of slate, it is probably sufficiently segregated from the lime to prevent trouble.

The importance of slag-formation temperature has long been recognized in non-ferrous metallurgy, but so far as I know it has never before been discussed in its relation to iron blast-furnace practice, nor has the importance of the mechanical association of the slag constituents been adequately recognized.

W. McA. JOHNSON, Hartford, Conn. (communication to the Secretary *):—I read Mr. Johnson's paper with pleasure and interest, and I give below a *résumé* of the construction and use of my slag-tester which it alluded to in the first foot-note.

A slag is essentially a complex silicate of several bases. Calcium oxide is one of the most prominent bases in iron blast-furnace practice, while in lead or copper blast-furnace work, ferrous oxide is the most prominent. In the process of cooling, the various silicates crystallize out similar to alloys, and as such, slags do not have a sharp melting-point.

In metallurgical work it is important that an accurate knowledge of the "running temperature" of the slag be had in order that a liquid slag be obtained under given metallurgical conditions. This is shown in the lead, iron, and copper furnace-practice especially in the first-named. Likewise in zinc retort work this knowledge is essential so as to prevent the formation of slags. The formation temperature varies greatly and is usually much below its melting-temperature, *i.e.*, the oxides of iron and calcium unite with silica in a dry way when intimately mixed and heated for a considerable interval of time. The question of time is an important point. Satisfactory data of this subject is not to be found in literature, for usually the uncertain "softening-point" was determined. Such results are not directly translatable into actual practice, unless subject to great corrections and metal reservations from the basis of practical knowledge.

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* Received Nov. 30, 1912.

While I was with the Orford Copper Co., in 1902, working on a small "nickel brick cupola," an opportunity was had to observe the process of slag-formation on the coke just as the cupola furnace was going out of blast at the end of the short weekly campaign. The observations were made through the small holes which appeared in the sides of the furnace.

While with the Lanyon Zinc Co., in 1903, I found that Acheson graphite was a most admirable material for making metallurgical research work. In that year an apparatus for the testing of the reduction-temperature of metallic oxides,[4] and a similar apparatus for the decomposition temperature of metallic sulphides was devised. On the basis of this varied experience, when with the Lungwitz Reduction Co., at the instance of Mr. Fred W. Gordon, chief consulting engineer, the "slag-tester" described here was developed for the purpose of making determinations of "slag-running temperatures" that were accurate. The apparatus as designed proved neat and serviceable, and gave results directly applicable to large scale work.

Round bars of Acheson graphite 4 in. in diameter were cut in 3.5 in lengths, with a narrow slot $\frac{5}{8}$ in. wide and 1.5 in. high cut in the bottom. A machined drilled 1 in. hole was made in the top to the depth of 1 in. A $\frac{7}{32}$ in. hole was continued to the narrow slot.

A horizontal 1 in. hole was drilled above the line of the lower slot so as to receive a Le Chatelier pyrometer at a point as near as possible to the $\frac{7}{32}$ in. "dripping-hole." The upper hole was filled with small proportions of the formed slag or slag-making materials. The entire apparatus was placed in a special oil-fired experimental reverberatory furnace, the hearth of which was covered with a layer of sand, and the pyrometer was inserted through a hole in one side of the furnace.

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[4] *Transactions of the American Electrochemical Society*, vol. v., p. 211 (April, 1904).

When the slag reached the "dripping-point," and dropped or ran through the slot, depending on its viscosity, it could be viewed directly through the slot by means of an observing-hole in the side of the furnace opposite the pyrometer-hole, and its dripping- or running-point determined. The accuracy of the results was increased by the fact that the pyrometer couple was so close to the slag at this critical temperature. Even though slag is a poor conductor of heat, Acheson graphite is so good a one that the temperature gradient is small.

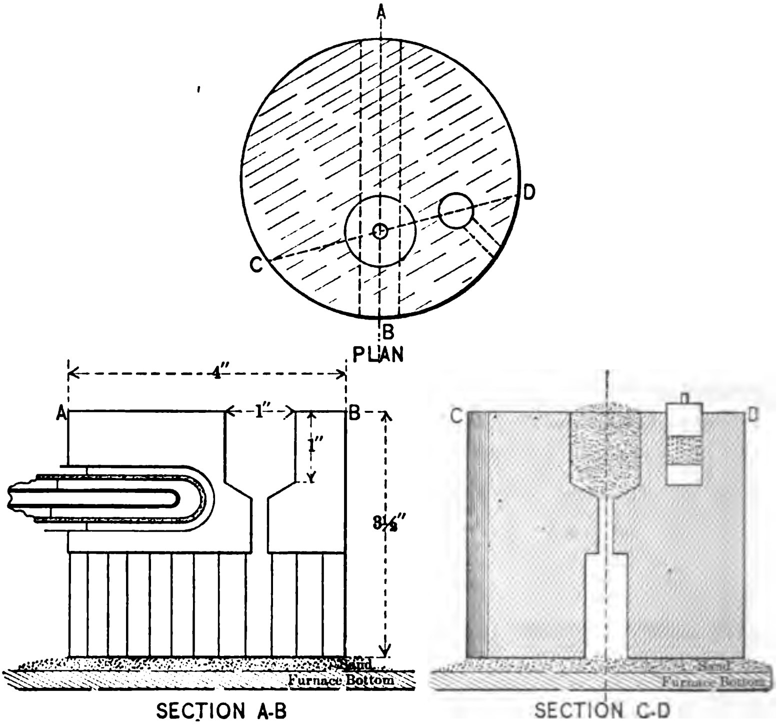

Fig. 1.—Slag-Tester. Plan and Sections.

A check on the pyrometer was secured by boring a second hole 0.5 in. in diameter in the top, in which 5 g. of zinc were put. The zinc was covered with charcoal and the hole plugged with a whittled graphite plug. A pyrometer was inserted through this hole and checked at the boiling-point of zinc, 920° C., shown by the characteristic snaky flame of burning zinc vapor coming through the pyrometer hole.

A much larger apparatus was also made, see Fig. 1; in which the slag was allowed to drip through a column of granulated coke 3 in. in diameter and 4 in. high. In this manner we measured the rate of reduction of reducible oxides, FeO and ZnO. Another apparatus was designed looking somewhat like an hour-glass, in which it was proposed to measure the rate of flow of slags through a small aperture at different temperatures.

Acheson graphite is the only commercial material to be used in making this apparatus, as fire-clay would be fluxed by the slag, and in using iron this would be oxidized, and the iron oxide thus formed would also be fluxed by the slag.

Several score of typical slags were tested in the apparatus, but the data of the test cannot be published, as the work was done in a cursory and fragmentary way. The results, if published, would be open to criticism that no safe or firm deduction could be made therefrom. By the use of a mental process somewhat better than semi-intelligent guessing, the results can be used by me.

This description of the apparatus has been rewritten (the original appeared in *Electrochemical and Metallurgical Industry*, Vol. IV., p. 262 (July, 1906), in the hope that if the proposal be thought a worthy one, an extended investigation of slags in this slag-tester be made by some one that can properly and efficiently do such work, *e.g.*, a professor in a university or some one connected with national or State government department.

SUBJECT TO REVISION.

[TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS.]

Measurements and Relations of Hardness and Depth of Carbonization in Case-Hardened Steel.

Discussion of the paper of Mark A. Ammon, presented at the Cleveland meeting, October, 1912, and printed in *Bulletin* No. 70, October, 1912, pp. 1167 to 1 79.

J. E. Johnson, Jr., Ashland, Wis. :—Mr. Ammon's paper is of great value and it is not with any desire to reflect on it that I wish to make some comments on the use of the scleroscope.

We obtained one on trial some months ago and made very extensive tests with it, but without getting very satisfactory results. The readings on good car-wheels and bad ones were just about the same, but the factor in its use most likely to produce erroneous results is a variation in the thickness of the specimen being tested.

In order to find out something on this subject we cut a thin disk, about $\frac{1}{16}$ in. thick, off a brass rod; then one 0.25 in. thick, and another about 1 in. thick, put the surfaces of all in about the same condition and tested them with the scleroscope. The readings varied widely; the resilience of the anvil evidently entered to a very high degree on thin pieces. By putting a soft cushion between such a thin specimen and the anvil, and forcing it up against the tube, the softness of the cushion cut off the resilience of the anvil and the results were more like those of the thick specimens.

After a thickness of 1 in. was passed, the specimen seemed to have mass enough so that the resilience of the anvil no longer played an important part, and above that thickness results on different materials would be comparable, even though the thicknesses differed. But in order to get comparable results on thin specimens the thickness of those being tested should be the same. I should like to ask how the variations introduced in this way were overcome, so as to get strictly comparable results.

Mr. Ammon :—I have not had much experience in that, but the scleroscope is simply an indication of the surface hardening. You cut them on an anvil of any ordinary firm basis, and you could not detect any practical difference, but to take a piece $\frac{3}{16}$ or $\frac{1}{8}$ in. thick, I have not had any experience on those lines at all. But the scleroscope is simply to show the elasticity of the surface.

Bradley Stoughton, New York, N. Y.:—I presume the reason Mr. Ammon did not find any differences was because the material was harder or more resilient than the anvil. This subject has come up very prominently before the American Society for Testing Materials. Now, that society has a committee which studies the question of hardness, and it was admitted by Mr. Shore himself, the inventor of the scleroscope, that thin pieces did not give good results. I presume, however, that if the thin pieces are harder than the anvil, the anvil would not affect them; but if they are softer, in a case where the pieces are $\frac{3}{16}$ of an in. thick, the effect of the anvil would be shown.

I want to say that there is a very great deal of information on the question of carbonizing and case-hardening in general, and hardening tests, and I suppose 80 per cent. of it has never been made public. I think that the Peerless Motor Company, to which these men belong, ought to be thanked by us for allowing the results that have been given here to be published, because we sincerely hope it will form a nucleus around which we can gather a great deal more valuable information on this subject. Case-hardening is no doubt going through an important transitional stage, being improved almost daily; and anything that can be done to get the many results in the shops made public, is of very great value to the industry to which this matter is so important.

SUBJECT TO REVISION.

[TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS.]

Notes on Ruff's Carbon-Iron Equilibrium Diagram.

Discussion of the paper of Prof. Henry M. Howe, presented in abstract by Bradley Stoughton at the Cleveland meeting, October, 1912, and printed in *Bulletin No.* 71, November, 1912, pp. 1181 to 1227.

J. E. JOHNSON, JR., Ashland, Wis.:—This material is largely beyond the understanding of most of us practical men, but there are certain portions of that diagram that are of enormous importance. We have been conducting a very extensive investigation of the quality of charcoal-iron and of all iron, and we have found that this eutectic point here (indicating on chart) is a sort of balance point between the good irons and the bad ones, and I have some slides to show what practical results we have found depending on this point here on the diagram; and it would be a mistake for men who were concerned with operation and were getting practical results, to think that these things are of no importance. They are exceedingly hard to understand, and I confess that a great deal of them is beyond me, but there is no doubt that the quality of iron depends to an enormous extent on the location of the carbon content of the iron to the right and left of this point, and the position of that point is dependent upon the silicon, and I wish to make a plea to the practical and operating members of the Institute not to disregard this material on account of its being highly scientific, because I hope to show later that there are matters of the highest practical importance that depend directly upon certain portions of this diagram.

HENRY D. HIBBARD, Plainfield, N. J.:—I understand that washed metal, which is practically pure iron and carbon, does not solidify at one instant, as cast-iron does, but passes through a mushy stage. That has been an enigma as to how and why it did so, and possibly some light might be thrown on it in this connection.

BRADLEY STOUGHTON, New York, N. Y.:—If Wittorff's propositions are accepted, washed metal containing more than 4.1 per cent. of carbon would begin to solidify as liquid plus Fe_4C

along the line MN. This line is so very steep that the least bit of carbon above the eutectic ratio would make a very large amount of Fe_4C. So that, almost immediately after the washed metal began to freeze, this much solid separating out would give us a mixture of liquid and solid, and the iron would have to cool from 1,350° down to 1,150°; that is, through a range of 200° C. (which is equal to about 350° F.) before it would be entirely solid; and that would explain this mushy state—part liquid and part solid—through which it would pass.

R. H. **Sweetser**, Columbus, Ohio:—Is there any way of applying the information brought out in these charts to the fact that in making very low silicon basic iron there are times when the silicon will be down between 0.2 and 0.3 per cent. and the sulphur under 0.03, when the pig-iron will be gray, with a little "bloom" of carbon deposited on the pigs just as soon as they chill, and at other times pig-iron of that analysis would be white all the way through? Is there any particular application of that chart which would answer why it should be gray at one time and white at another?

Mr. Stoughton:—I presume Mr. Sweetser has taken into consideration the casting-temperature in each case.

Mr. Sweetser:—I do not know what the temperature is in either case.

Mr. Stoughton:—I should suppose then that it is a case of casting-temperature or rate of cooling, because the higher the temperature at which it is cast, the more likely it is to be white, and the lower the temperature at which it is cast, the more likely it is to be gray. Where it is just in that intermediate stage where it may be either one, a slight difference in the casting-temperature might push it to either side of the line.

Mr. Sweetser:—Which side of the line have we got to go to get that gray iron? Can you tell me that from the chart?

Mr. Stoughton:—With 3 per cent. of silicon you ought to get gray iron on the right-hand side of the eutectic iron. The eutectic line, with perhaps 0.3 per cent. of silicon and phosphorus under 0.2, might be under 2.9 per cent.; it might be somewhat lower than that. If you had less than the eutectic amount of total carbon, it would be white, and with more it would be gray.

Mr. Sweetser:—3.9 per cent. of carbon?

Mr. Stoughton:—I am simply stating that from such investigation as I have made. We have not enough to be certain.

Mr. Sweetser:—I have noticed that in using Pocahontas coke it is possible to make an iron with a very low silicon and still get a gray iron even when cast in chills. And there have been times when the iron would be chilled all the way through to white iron. If there is any way to connect that chart with the question of just what side of the eutectic line the white iron is, it would be a great help to practical furnace-men. And I do want to say here that although that chart is very complicated to me, I hope some time to learn something about it. A few years ago Dr. Howe gave a tri-axial chart which seemed as complicated to me as this one, but I have since learned that it can be of use in blast-furnace practice; and if this new chart is practical, then we are going a good ways ahead.

Mr. Johnson:—If Mr. Sweetser will make carbon determinations of the white iron and the gray, I think he will find where the difference comes in. We are more or less familiar with that phenomenon, and frequently get irons almost totally deficient in chill, although the silicon is such that they should have a chill. The thing we have found in regard to them is that they are high-carbon irons, but incidentally they are physically weak and rotten iron.

James F. Kemp, New York, N. Y.:—It may be not entirely without interest to the members of the Institute to know that geologists are working along very similar lines to those set forth in this diagram. This term "eutectic" has become very familiar to us. In analyzing molten rocks we go through processes very similar to those which are employed in the study of alloys. There are many surprising and very close parallels between the structures of the igneous rocks which we find in the field and the structures brought out by the metallographic investigation of alloys.

SUBJECT TO REVISION.

[TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS.]

Notes on Titanium, and the Cleansing-Effect of Titanium on Cast-Iron.

Discussion of the paper of Bradley Stoughton, presented at the Cleveland meeting, October, 1912, and printed in *Bulletin No.* 71, November, 1912, pp. 1245 to 1275.

J. E. JOHNSON, JR., Ashland, Wis.:—Is it true that titanium reduces the chill in chilling iron? If so, as I think Mr. Stoughton has stated elsewhere, then I do not see how titanium is beneficial in car-wheel mixtures. One of the most difficult things to do is to get a proper chill and a proper degree of toughness, and the goodness of an iron of the given composition for charcoal-iron purposes is denoted by the chill which it will take as compared with its composition. If titanium reduces that chill, then it does not seem to me as good for car-wheels as its absence would be, and I should like to have it explained what effect titanium has on the chill of a given iron.

MR. STOUGHTON: It is true that titanium reduces the chill of iron quite noticeably, yet, at the same time, if I am correctly informed, 90 per cent. of all chilled-iron rolls are made to-day with the titanium treatment, and a great many car-wheels, but one has to make changes in the silicon added and also, if I am correctly informed, in the manganese. The advantage of titanium is that when the proper changes are made, so that we get the same depth of chill with the titanium as we otherwise would without, the chill is much harder than before, and also the body of the roll or wheel is tougher and somewhat stronger, especially under shock.

JAMES F. KEMP, New York, N. Y.:—This subject is obviously one of great importance to the iron industry, in that if we can find a method of using titaniferous ores, we will prolong the life of our iron reserves. Fifteen or more years ago, primarily from the study of the ores in the Adirondacks, I became very much interested in this type of deposit.* We know that there are many millions of tons of them in that section of the country, and to the north of it in Canada. They are also abundant in Minnesota north of Lake Superior; in Wyoming, Colorado, Virginia and North Carolina, and if we can bring them into actual use they will add greatly to the iron-ore resources of this continent. We also know that in the titaniferous ores vanadium is at home. About fifteen years ago, in platting curves illustrating the composition of the ores, so far as accurate detailed analyses could be obtained, I found that there was a very peculiar sympathy between the curves of chromium and vanadium. I do not think any of us know the form in which vanadium exists in these ores. But the vanadium and the chromium curves went up and down together in very close sympathy. I dare not say that these two elements were combined in a rare mineral, but that there was this sympathy is an undeniable fact. It is quite possible that some new metallurgical improvements will bring within reach of the furnacemen ore that now is avoided, and has been avoided in the past as it it were a plague. At present the titanium is chiefly derived from rutile.

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* Titaniferous Iron Ores of the Adirondacks, *Nineteenth Annual Report, U. S. Geological Survey*, Pt. III., pp. 377–422 (1899). A Brief Review of the Titaniferous Magnetites, *School of Mines Quarterly*, vol. 20, pp. 323–356; vol. 21, pp. 56–65 (1899).

SUBJECT TO REVISION.

[TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS.]

Blowing-In a Blast-Furnace.

Discussion of the paper of R. H. Sweetser, presented at the Cleveland meeting, October, 1912, and printed in *Bulletin No.* 71, November, 1912, pp. 1327 to 1334.

J. E. JOHNSON, JR., Ashland, Wis.:—There is one method of bringing down the gas which Mr. Sweetser has not mentioned, and which is very useful. John W. Dougherty, now of the Crucible Steel Co. of America, is entitled to the credit for the common knowledge of a plan. This consists in turning steam into the downcomer or dust-catcher before lighting the furnace, so as to form a steam cushion between the air and the gas. The danger of course comes, not from the air alone or from the gas alone, but from the mixture of the two, which is produced when the gas first begins to come through. The steam cushion between the air and the gas prevents the possibility of an explosive mixture being formed. Mr. Dougherty has a patent on this process, but I believe it is established that the process was in use elsewhere some two years before his patent was applied for, and therefore the patent is of doubtful validity, which is unfortunate, because Mr. Dougherty undoubtedly evolved the idea independently of its previous use.

In blowing-in a furnace, I think it good practice to speed the engine up for a few minutes to a considerably higher speed than you expect to use after the gas is brought down. There is at first a very slow action, due to the great quantity of voids in the furnace to be filled up, and the progress of the gas is exceedingly slow, if the slow blowing which is suitable for blowing-in be used from the very beginning. By running the engine at 50 or even 100 per cent. greater than the blowing-in speed for a few minutes, the gas is brought through promptly, and the dangerous period very greatly shortened. After the gas is brought down and safely lighted, the engine can be put down to its proper speed. This need not be more than 10 or 15 min. I lit a furnace about a month ago, following this practice, and the gas was down and lighted at all burners in about 10 min. from the time the wind was turned on, without a sign of a puff of any kind. We used the steam cushion, and also led the gas to the far end of the flue, opening the boiler- and stove-burners after the thick white first gas had pushed the steam out of the opening.

In connection with the stopping of the tap-hole, a very good plan is to build a considerable pile of clay, mixed with coke- or coal-braize, over the tap-hole, and extending back well into the hearth. The pipe described by Mr. Sweetser is arranged to project through this into the furnace. When the tap-hole comes to be opened, it is easy to drill through this soft mass to the skull which the heat of the hearth bakes on its surface, and a bar can then easily be driven through this skull.

This plan may save a great deal of trouble when it comes to opening the iron-notch. The principle is the same as that of the cinder-notch which projects into the molten matter in the hearth. It is also the same as the deep hole kept by the aid of the steam-gun after the furnace is in operation.

Mr. Sweetser's plan for pulling the pipe out of the tap-hole when it is desired to close the hole, is quite the best that I have heard of.

SUBJECT TO REVISION.

[TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS.]

Notes on the Case-Hardening of Special Steels.

Discussion of the paper of Prof. Albert Sauveur and G. A. Reinhardt, presented at the Cleveland meeting, October, 1912, and printed in *Bulletin No.* 71, November, 1912, pp. 1335 to 1341.

ROBERT R. ABBOTT, Cleveland, Ohio:—The possibilities of the commercial application of the facts brought out in this paper have been the subject of many experiments by me during the past four years. It will be well to call attention to several facts having a commercial bearing upon the problem:

(1) The lower critical temperature (Ac_1) of a 3.5 per cent. nickel-steel of low carbon (say 0.20 per cent.) content is about 710° C. This is not lowered to any extent by increasing its carbon content by carbonizing.. By this I mean that a steel with Ac_1 occurring at 710°, must, after carbonizing, be heated above this temperature before quenching in order to make it hard. This is not contrary to Guillet's diagram, although at first thought it may seem so, due to a confusion of the Ac_1 and the Ar_1 points. His diagram merely gives the physical condition of the steel upon slow cooling. Now a steel of 1.20 per cent. of carbon and 3 per cent. of nickel (point "B" of the diagram) may be martensitic upon slow cooling from three causes:

(*a*) Its Ar_1 point is below the normal atmospheric temperatures.

(*b*) The time element is so important that the change from martensite to pearlite and cementite does not have time to occur even with this slower rate of cooling. We of course know that nickel has the tendency of retarding this change.

(*c*) A combination of the above two causes.

(2) With increasing nickel content the ability of steel to absorb carbon decreases.

(3) Many carbonized nickel-steels show a martensitic structure upon air-cooling, and yet are so soft that they are of no commercial value.

SUBJECT TO REVISION.

[TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS.]

The Concentration of Iron-Ores.

Discussion of the paper of N. V. Hansell, presented at the Cleveland Meeting, October, 1912, and published in *Bulletin* No. 72, December, 1912. (This issue.)

E. G. Spilsbury, New York, N. Y.:—I notice that Mr. Hansell has spoken in his paper about the work under the Goltra patents which is now being carried on at Waukon, Iowa. I had occasion a few months ago to visit that plant and was surprised to see what they were doing. The ore-body is practically a clay bed containing nodules of hematite of all sizes from large boulders down to small grains. It is mined by steam-shovel, and the iron content of the whole mass as mined varies from 22 to 26 per cent.

The result of the use of the Goltra process in the dry concentration of this ore is a material running from 55 to 59 per cent. of iron, and physically in a condition remarkably desirable for the blast-furnace. The process is rather a curious one; the ore delivered from the shovel is brought down to the mill, and after screening so as to take out the very large boulders, which go to the crusher, the whole mass is delivered into the horizontal revolving-kilns 155 ft. long and some 10 ft. in diameter. The secret of the success depends a great deal on the preliminary heating, which must be so graduated as to prevent the balling-up of the clay. The endeavor is to dry the material gradually in such a way that the clay is left in a finely divided state. The blast is introduced at the discharge end of the rotary kiln and actually blows out all of the clay-dust with a comparatively small percentage of iron. The ore as it progresses towards the discharge end is gradually raised to a very high heat and as it comes from the kiln is plunged into water and then runs through revolving-screens. The oversize from these screens is delivered to a long picking-belt. The process of roasting and plunging in water shows up the chert and quartz very thoroughly so that they are easily picked out. The fines passing through the screens then go to a second rotary kiln, which is hermetically sealed, and into which is admitted a certain amount of oil for fuel. The result of this second heating in a reducing atmosphere is the production of a magnetic oxide. This product then goes to the magnetic separators, which are of the Ball-Norton type.

While all the difficulties from the practical operation of the process had at the time of my visit not been entirely overcome, I consider that the process has the elements of very great success, and if the figures claimed for the cost of treatment do not exceed 30 cents per ton of concentrated ore, it ought to be a very profitable one.

At the time of my visit no treatment was being given to the flue-dust resulting from the blowing-out of the fine clay, but experiments had been made on a pneumatic concentrator on this material which had given very good results, and with such concentrators introduced, the element of loss of ore in the treatment will be reduced to a minimum. The process is certainly a very novel one and shows great ingenuity on the part of the inventor.

PHILIP N. MOORE, St. Louis, Mo.:—It is interesting to note the recurrence, with the use of better mechanical appliances, of an idea carried out years ago. In the 70's J. H. Hillman, recently deceased at Pittsburg, then superintendent of the Center Furnace of the Hillman Co., in western Kentucky, blew the siliceous dust from roasted brown ores by the use of a fan, thereby materially enriching them. These ores were roasted in open heaps, and, as the district has a considerable rainfall, the process was only of value during the dry season.

J. E. JOHNSON, JR., Ashland, Wis.:—What advantage has this roasting-process over the old-fashioned process of log washing which has been practiced in the eastern Pennsylvania and southern regions for many years, and with which I was at one time very familiar? We could carry that on and take out from the ore everything that was not attached to the ore itself, and we could do it for about 7 cents a ton; I think this process mentioned here would go closer to $1 a ton, and I do not see what is going to pay for the difference, or where there is room for much better recovery than we got, because we discarded very little. The loss in weight was about one-third, and in many cases that was so low in iron that it did not pay to treat it any further. While this process is interesting, I do not see where its commercial value comes in as compared with the old-fashioned method of washing.

Ms. Spilsbury:—They have been using the washing-process for a great many years, and obtained material which was probably from 46 to 48 per cent. of iron, whereas under this new method they are producing material of 66 per cent. iron content, and it is in a condition desirable for blast-furnaces.

Mr. Johnson:—Most of that increase, however, came from the expulsion of the combined water, which is done automatically in the blast-furnace.

E. V. D'Invilliers, Philadelphia, Pa.:—I do not think that with the use of the old washing-method they have ever had anything as lean as 25 per cent. of iron to treat. Might there not be some economies in the washer to work materials of that kind. The ores were lean, I do not think that they were ever washed much better than from 42 to 46 per cent. of iron in the old days, with the attempt made to carry the process through jigging-methods to get rid of the free quartz.

Mr. Johnson:—There are plenty of places in Alabama, Virginia, and Georgia, where they expect to handle 12 tons of dirt to get a ton of ore, and it is done right along as a commercial proposition. These ores are frequently raised to 46 per cent. of iron, and I know of a case in which the iron was raised to more than 50 per cent.

[TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS.]

The Action of Various Commercial Carbonizing-Materials.

Discussion of the paper of Robert R. Abbott, presented at the Cleveland meeting, and printed in *Bulletin* No. 72, December, 1912 (this issue).

H. D. HIBBARD, Plainfield, N. J.:—Were any of these carbonizing-materials gaseous?

MR. ABBOTT:—None of them were.

MR. HIBBARD:—It is regrettable that the gases were not included. From some indications gas carbonizing is leading at present, and I would like to know the performance of gaseous as compared with solid carbonizers.

MR. ABBOTT:—Straight gases were not included because the materials which I used gave up the various gases which are being used in gas carbonizing. In other words, I generated in the materials the gases which are sometimes used as compressed gases.

MR. HIBBARD:—It is considered uncertain what gases and what amount of them are given out, and the temperatures at which they are set free from these materials. I have heard it claimed that the temperatures at which case-hardening proceeds rapidly are above those at which the gases are given out, and so by the time that the gases are pretty well expelled from the material, case-hardening is hardly ready to begin, and therefore the results have been non-uniform and unsatisfactory.

SUBJECT TO REVISION.

[TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS.]

Recent Developments in the Inspection of Steel Rails.

Discussion of the paper of Robert W. Hunt, presented at the Cleveland meeting, October, 1912, and published in *Bulletin No.* 72, December, 1912 (this issue).

H. D. Hibbard, Plainfield, N. J.:—What is the usual amount of discard from the top of the ingot? Does it often happen that additional discard is made in order to secure soundness of the top rail of the ingot?

Mr. Hunt:—There is no uniform practice. The generally accepted one, without its being written in the contract, is that the discard shall amount to about 9 per cent. Many of the specifications provide that it shall continue until seemingly sound steel is reached in the bloom. A few, but very few, contracts have a specific discard of from 20 to 25 per cent., but that is exceptional, and for that an additional charge is made by the manufacturer. That is one of the questions which is now agitating the purchasers of rails—that is, the railway people—as to what future steps they shall take. There is a very strong tendency towards insisting upon at least one-third of the ingot not being used for rails. One railway company purchased a large quantity of rails where, without its being so stated in the contract, it was understood that the top bloom would not be given to them. The manufacturers, fortunately for them, had two rail-mills, and they diverted the top bloom to the other mill, where it was rolled into rails of other sections. Who got them I am not prepared to say.

Wm. H. Blauvelt, Syracuse, N. Y:—I should like to ask Mr. Hunt whether the same organized effort is being made to improve the uniformity of the steel in the ingot before it passes to the rolls, among the steel-makers as a whole, as he has described in this treatment of the steel by the rolling-process.

Mr. Hunt:—This form of inspection covers the metal from the time it is tapped from the furnace or poured from the converter; its action in the ladle; what additions are made to it, and if there is any variation from the works' accepted good practice, as, if the slag comes ahead of the steel in tapping; if the ladle-stopper does not work well, and it is a badly-poured heat; if cracked ingot-molds are used; if some of the ingots are stickers, etc.—in fact, the whole history of the process becomes a matter of record, without any interference on the part of the inspector. The result has been, I think I am quite safe in saying, that there have been a great number of heats (I will not try to give accurate figures) that have not been rolled into rails, which, under old conditions, would have gone. They have been diverted to other purposes, or put aside, and not applied to that particular rail-contract—and I say this without any disrespect to the steel-makers, I mean the managers of works. No manager can be in all parts of his works at all times and see everything, and as long as the men are paid by the ton, it is human nature to try to get the product past.

Frederick Laist, Anaconda, Mont.:—I read somewhere that about 2 per cent. of copper was occasionally added to rail-steel for the purpose of toughening it. Has this been done, and, if so, has it given successful results?

Mr. Hunt:—The Chicago, Milwaukee & St. Paul railroad had 5,000 tons of rails made from steel to which copper was added; it was expected to secure about 0.5 per cent. of copper in the rail; that is, they were seeking to obtain about 0.5 per cent. of copper in the metal. These rails were made by the Illinois Steel Co. The manufacturing results were entirely satisfactory. The copper was added to the steel in the casting-ladle in the form of copper wire, so as to get small individual pieces. The idea originated with the railway; in fact, I might say that it was due to E. W. McKenna, Vice-President of the Chicago, Milwaukee & St. Paul railroad, who has always taken a great interest in steel rails and was the inventor and introducer of the McKenna process, so called, for re-rolling worn rails of large sections and still keeping them to standard sections, which practice has been successfully working now for some 12 or 15 years. Mr. McKenna's idea was based upon the satisfactory results which had been obtained from the rails made by the Bessemer process from ores from Cornwall, Pa., which contained a percentage of copper that yielded about 0.5 per cent. in the steel; the analyses of the rails showed about 0.5 per cent. of copper, so Mr. McKenna thought that they would see what putting it in the steel in the above-described way would accomplish. Of course, what it will accomplish, if any thing, remains to be seen.

ELWOOD HAYNES, Kokomo, Ind.:—The inspection of the rails, as I understand it, applies to the finished metal, ready for making into rails, and then to the rail itself and to the method of treatment. It seems to me that back of that the process of making the steel is fully as important as the inspection of the ingot and the rail after it is worked out. The introduction of titanium or other purifying agents into the steel, it seems to me, would have fully as much effect as the proper treatment of the steel in rolling it into rails. In the automobile business I know that we are very particular to use, in certain cases where the material is subjected to vibration, either a nickel-steel or a vanadium-steel. Of course those materials would be too expensive for steel rails. I have made a great many personal tests of material under shock and under vibration. When perfectly good steel is placed under vibration a breaking-up of the fibers takes place after a little while, usually after 40,000 or 50,000 vibrations of a certain amplitude, as is well known, of course, to engineers, and the apparently good piece of steel breaks. The physical quality or its general appearance apparently makes little difference. If it contains a certain amount of vanadium or nickel, the break does not occur until after millions of vibrations. It seems to me that the inspection should go back to the composition of the rail, which is fully as important as the treatment.

MR. HIBBARD:—When shearing off the usual amount discloses unsound metal, and the shearing is continued until sound metal is reached, who determines when this additional shearing shall take place and what it shall amount to?

MR. HUNT:—That is a point which I am glad you raised. It has led to perhaps as much dispute in regard to the working of this inspection as any one point. It is a very delicate situation, as to how the inspector shall act under those conditions. I am insisting that it is not the inspector's duty; or, rather, under the existing contracts, the manufacturer would have a right to object. If the inspector should say to the shearsman, "You must continue cutting that bloom off, you have not cut it enough," etc., the responsibility then would be put on the inspector, and the manufacturer relieved from it, and his workmen relieved—as the shearsman cut as long as the inspector told him to, etc. The inspector's relations are not quite that, and I do not think it would bring as good results as if he simply satisfied himself that a certain ingot of Heat No. so-and-so had been improperly sheared, and immediately reported that knowledge to the foreman in charge of that department in somewhat this form: "Your shearsman did not properly shear that bloom or those blooms." Such a reported condition follows that bloom into the rail; that rail has that individual history attached to it, and it follows it into service, and the knowledge of such notation and report will, in my judgment, result in the workmen being more careful than if they could say, "Well, the inspector decided the bloom was sheared enough," or, "I cut it as far as the inspector wanted," etc. Proper shearing is often a difficult point to determine. As every steel-maker knows, the movement of the shear, its physical action on the steel, is very much like cutting a piece of cheese with a knife; it will carry the metal down and perhaps conceal a porous condition. A very pronounced case of unsoundness can be readily detected, but that condition is not always so evident.

R. H. **Sweetser**, Columbus, Ohio:—What is the limit of phosphorus in Bessemer rails, and is the upper limit more closely adhered to than it has been for the last several years in the United States?

Mr. Hunt:—Not to exceed 0.10 per cent. is the universal Bessemer specification. It seldom gets above 0.096 per cent.; in fact, I think the general average has been during the last few years perhaps a little lower than it was preceding that time. As a rough reason for it, without having data to back the opinion, I believe that the large consumption—or the large percentage of diversion, if I may so put it, of rail orders to the open-hearth process has relieved the strain on the Bessemer ores, so that those mills which were running on Bessemer rails have had less trouble in obtaining ores giving them phosphorus well within the 0.10 per cent. limit than was the case a few years ago when all the rails were made by the Bessemer process. At all events, I know the current reports are showing an average of somewhat lower phosphorus.

BULLETIN OF THE AMERICAN INSTITUTE OF MINING ENGINEERS.

INDEX OF TITLES AND AUTHORS.

PAPERS FOR THE YEAR 1912.

BULLETIN, 1912.

No. 61, JANUARY	(lxi)	No. 67, JULY	(lxvii)
No. 62, FEBRUARY	(lxii)	No. 68, AUGUST	(lxviii)
No. 63, MARCH	(lxiii)	No. 69, SEPTEMBER	(lxix)
No. 64, APRIL	(lxiv)	No. 70, OCTOBER	(lxx)
No. 65, MAY	(lxv)	No. 71, NOVEMBER	(lxxi)
No. 66, JUNE	(lxvi)	No. 72, DECEMBER	(lxxii)

American Institute of Mining Engineers,
29 West 39th St., New York, N. Y.

INDEX OF TITLES AND AUTHORS FOR THE YEAR 1912.

Abbott, Robert R. The Action of Various Commercial Carbonizing-Materials, lxxii, 1407, 1565.

Abbott, Robert R. *Discussion* of Notes on the Case-Hardening of Special Steels, lxxii, 1563.

Abrasion and Dust-Losses in Ore-Drying. By Carl F. Dietz and Dyke V. Keedy, lxvii, 717.

Action of Various Commercial Carbonizing-Materials. By Robert R. Abbott, lxxii, 1407.

Agglomeration of Fine Materials. By Walter S. Landis, lxv, 527.

Alaska Coal-Land Problems. By H. Foster Bain (and *Discussion*), lxviii, 811.

Aldrich, T. H., Jr. Electrolytic Oxygen in Cyanide Solutions, lxii, 177.

Alundum Extraction-Thimble Used in the Determination of Copper. By L. W. Bahney, lxxi, 1343.

Ammon, Mark A. Measurements and Relations of Hardness and Depth of Carbonization in Case-Hardened Steel, lxx, 1167; lxxii, 1550.

Bahney, L. W. The Alundum Extraction-Thimble Used in the Determination of Copper, lxxi, 1343.

Bain, H. Foster. Alaska Coal-Land Problems, lxviii, 811.

Bain, H. Foster. *Discussion* of Notes on the Laramie Tunnel, lxvi, 659.

Bearing of the Theories of the Origin of Magnetic Iron-Ores on Their Possible Extent. By Frank L. Nason, lxvii, 695.

Black Mountain Coal-District, Kentucky. By J. B. Dilworth, lxii, 149.

Blauvelt, W. H. The Manufacture of Coke, lxxi, 1299.

Blauvelt, William H. *Discussion* on Recent Developments in the Inspection of Steel Rails, lxxii, 1567.

Blowing-In a Blast-Furnace. By R. H. Sweetser, lxxi, 1327.

Bordeaux, A. F. J. Present Conditions of Mining in the District of Vladivostok, Siberia, lxx, 1025.

BRANNER, J. C. An Early Discovery of Fullers' Earth in Arkansas, lxvii, 747.

Briquetting of Iron-Ores. By N. V. HANSELL, lxv, 557.

BRUNTON, DAVID W. Notes on the Laramie Tunnel, lxiv, 357.

CAMPBELL, WILLIAM. The Microstructure of Iron and Steel, lxxii, 1357.

CAMPBELL, WILLIAM. Notes on the Metallography of Alloys, lxxii, 1381.

Centrifugal Machines for Ore-Grading and Ore-Concentrating. By GODFREY T. VIVIAN, lxx, 1103.

Clinton Iron-Ore Deposits in Kentucky and Tennessee. By S. WHINERY, lxx, 1057.

COLLINS, G. A. Efficiency-Engineering Applied to Mining, lxix, 995.

Concentration of Iron-Ores. By N. V. HANSELL, lxxii, 1497.

Concise Method of Showing Ore-Reserves. By N. H. EMMONS, lxvi, 603.

Concise Method of Showing Ore-Reserves. *Discussion* by E. W. KING, JOHN D. IRVING, CHARLES KIRCHHOFF, WILLIAM KELLY, N. S. KEITH, lxvi, 608.

Constitution and Melting-Points of a Series of Copper-Slags. By CHARLES H. FULTON, lxxii, 1457.

Contribution to the Study of Pre-Cambrian Rocks of the Harney Peak District of South Dakota. By GORDON S. DUNCAN, lxvii, 751.

COTTRELL, F. G. Electrical Fume-Precipitation, lxvii, 667.

Cyanide-Plant at the Treadwell Mines, Alaska. By W. P. LASS, lxii, 183.

DANIELS, JOSEPH. The Fritz Engineering and the Coxe Mining Laboratories at Lehigh University, lxii, 227.

Decomposition of Metallic Sulphates at Elevated Temperatures in a Current of Dry Air. By H. O. HOFMAN and W. WANJUKOW, lxix, 889.

DENNIS, FRANCIS J. Examination of Dredging-Properties, lxiv, 407.

Development of the Parkes Process in the United States. By ERNST F. EURICH, lxxii, 1531.

DEWEY, FREDERIC P. The Direct Determination of Small Amounts of Platinum in Ores and Bullion, lxiv, 439.

DEWEY, F. P. The Sampling of Gold-Bullion, lxx, 1075.

DIETZ, CARL F., and KEEDY, DYKE V. Abrasion and Dust-Losses in Ore-Drying, lxvii, 717.

DILWORTH, J. B. The Black Mountain Coal-District, Kentucky, lxii, 149.

D'INVILLIERS, E. V. *Discussion* of The Concentration of Iron-Ores, lxxii, 1565.

Direct Determination of Small Amounts of Platinum in Ores and Bullion. By FREDERIC P. DEWEY, lxiv, 439.

DUMBLE, E. T. The Occurrence of Gold in the Oocene Deposits of Texas, lxx, 1021.

DUNCAN, GORDON S. Contribution to the Study of Pre-Cambrian Rocks of the Harney Peak District of South Dakota, lxvii, 751.

Early Discovery of Fullers' Earth in Arkansas. By J. C. BRANNER, lxvii, 747.

Effect of Alumina in Blast-Furnace Slags. By J. E. JOHNSON, lxx, 1123.

Efficiency-Engineering Applied to Mining. By G. A. COLLINS, lxix, 995.

EILERS, ANTON. *Discussion* of Flameless Combustion, lxix, 963.

EILERS, ANTON. *Discussion* of The Refining of Blister-Copper lxvii, 767.

Electric Hoist at the Hecla Mine. By E. M. MURPHY, lxix, 977.

Electrical Fume-Precipitation. By F. G. COTTRELL, lxvii, 667.

Electrolytic Oxygen in Cyanide Solutions. By T. H. ALDRICH, JR., lxii, 177.

Electrostatic Concentration or Separation of Ores. By HENRY A. WENTWORTH, lxvi, 633.

ELLIOTT, ARTHUR H. *Discussion* of Flameless Combustion, lxix, 963.

ELLIS, CARLETON. Flameless Combustion, lxix, 945.

EMMONS, N. H. A Concise Method of Showing Ore-Reserves, lxvi, 603.

EMRICH, HORACE H. The Refining of Blister-Copper, lxvi, 611.

EURICH, ERNST F. The Development of the Parkes Process in the United States, lxxii, 1531.

Examination of Dredging-Properties. By FRANCIS J. DENNIS, lxiv, 407.

Excursion to Hawaii and Japan. By R. W. RAYMOND, lxi, 1.

Experience in the Use of Water-Power. By C. M. MYRICK, lxx, 1071.

Fires in Metalliferous Mines. By GEORGE J. YOUNG, lxx, 1133.

Flameless Combustion. By CARLETON ELLIS, lxix, 945.

Flameless Combustion. *Discussion* by ARTHUR H. ELLIOTT, C. A. MEISSNER, ANTON EILERS, WILLIAM KENT, CHARLES E. LUCKE, lxix, 963.

FORBES, D. L. H. The Treatment of Complex Silver-Ore at the Lucky-Tiger Mine, El Tigre, Sonora, Mexico, lxviii, 769.

FOSTER, RUFUS J. *Discussion* of Mine Caves Under the City of Scranton, lxiv, 443.

Fritz Engineering and Coxe Mining Laboratories of Lehigh University. By JOSEPH DANIELS, lxii, 227.

FULTON, CHARLES H. The Constitution and Melting-Points of a Series of Copper-Slags, lxxii, 1457.

FULTON, C. H. *Discussion* of The Effect of Alumina in Blast-Furnace Slags, lxxii, 1542.

GAMBA, FORTUNATO PEREIRA. Gold-Mines in Southern Colombia, lxviii, 853.

Geology and Ore-Deposits of the Silverbell Mining-District, Arizona. By CHARLES A. STEWART, lxv, 455.

Geology of Harrison Gulch, in Shasta County, California. By H. E. KRAMM, lxvii, 709.

Geology of the Tonopah Mining-District. By AUGUSTUS LOCKE, lxii, 217.

Gold-Mines in Southern Colombia. By FORTUNATO PEREIRA GAMBA, lxviii, 853.

Gold-Production in California. By CHARLES G. YALE, lxiii, 261.

Graphic Chart for the Solution of D'Arcy's Formula for the Transmission of Compressed Air in Pipes, lxxii, 1519.

HAMPTON, WILLIAM H. *Discussion* of Alaska Coal-Land Problems, lxviii, 811.

HANSELL, N. V. The Briquetting of Iron-Ores, lxv, 557.

HANSELL, N. V. The Concentration of Iron-Ores, lxxii, 1497.

HAYNES, ELWOOD. *Discussion* on Recent Developments in the Inspection of Steel Rails, lxxii, 1571.

HERZ, NATHANIEL. A Graphic Chart for the Solution of D'Arcy's Formula for the Transmission of Compressed Air in Pipes, lxxii, 1519.

HIBBARD, HENRY D. *Discussion* of Notes on Ruff's Carbon-Iron Equilibrium Diagram, lxxii, 1555.

HIBBARD, HENRY D. *Discussion* on Recent Developments in the Inspection of Steel Rails, lxxii, 1571.

HIBBARD, HENRY D. *Discussion* of The Action of Various Commercial Carbonizing Materials, lxxii, 1569.

HOFMAN, H. O., and WANJUKOW, W. The Decomposition of Metallic Sulphates at Elevated Temperatures in a Current of Dry Air, lxix, 889.

HOWE, HENRY M. Notes on Ruff's Carbon-Iron Equilibrium Diagram, lxxi, 1181.

HUNT, ROBERT W. Recent Developments in the Inspection of Steel Rails, lxxii, 1487, 1567.

IRVING, JOHN D. *Discussion* of A Concise Method of Showing Ore-Reserves, lxvi, 608.

James Diagonal-Plane Slimer. By S. ARTHUR KROM, lxvi, 649.

JANIN, CHARLES. Present-Day Problems in California Gold-Dredging, lxiii, 241.

JENNINGS, E. P. A Titaniferous Iron-Ore Deposit in Boulder County, Colo., lxx, 1045.

JOHNSON, J. E., JR. The Effect of Alumina in Blast-Furnace Slags, lxx, 1123; lxxii, 1541.

JOHNSON, J. E., JR. *Discussion* of Blowing-In a Blast-Furnace, lxxii, 1561.

JOHNSON, J. E., JR. *Discussion* of Measurements and Relations of Hardness and Depth of Carbonization in Case-Hardened Steel, lxxii, 1553

JOHNSON, J. E., JR. *Discussion* of Notes on Ruff's Carbon-Iron Equilibrium Diagram, lxxii, 1555.

JOHNSON, J. E., JR. *Discussion* of Notes on Titanium, and the Cleansing Effect of Titanium on Cast-Iron, lxxii, 1559.

KEEDY, DYKE V., and KARL F. DIETZ. Abrasion and Dust-Losses in Ore-Drying, lxvii, 717.

KEITH, N. S. *Discussion* of A Concise Method of Showing Ore-Reserves, lxvi, 609.

KELLER, EDWARD. Solubility in Nitric Acid of Gold Contained in Certain Copper-Alloys (Copper Bullions), lxvii, 681.

KELLY, WILLIAM. *Discussion* of a Concise Method of Showing Ore-Reserves, lxvi, 609.

KEMP, JAMES F. *Discussion* of The Effect of Alumina in Blast Furnace Slags, lxxii, 1541.

KEMP, JAMES F. *Discussion* of Notes on Ruff's Carbon-Iron Equilibrium Diagram, lxxii, 1555.

KEMP, JAMES F. *Discussion* of Notes on Titanium, and the Cleansing Effect of Titanium on Cast-Iron, lxxii, 1559.

KENT, WILLIAM. *Discussion* of Flameless Combustion, lxix, 964.

KING, E. W. *Discussion* of A Concise Method of Showing Ore-Reserves, lxvi, 608.

KIRCHHOFF, CHARLES. *Discussion* of A Concise Method of Showing Ore-Reserves, lxvi, 608.

KLUGH, B. G. The Sintering of Fine Iron-Bearing Materials by the Dwight & Lloyd Process, lxv, 507.

KRAMM, H. E. Geology of Harrison Gulch, in Shasta County, California, lxvii, 709.

KROM, S. ARTHUR. The James Diagonal-Plane Slimer, lxvi 649.

KROM, S. ARTHUR. *Discussion* of Progress in Roll-Crushing, lxvi, 595.

LAIST, FREDERICK. *Discussion* on Recent Developments in the Inspection of Steel Rails, lxxii, 1571.

LANDIS, WALTER S. Agglomeration of Fine Materials, lxv, 527.

LASS, W. P. The Cyanide-Plant at the Treadwell Mines, Alaska, lxii, 183.

Laws of Igneous Emanation Pressure. By BLAMEY STEVENS, lxiv, 411.

LEDOUX, ALBERT R. *Discussion* of the Refining of Blister-Copper, lxvi, 629.

LIVINGSTON. D. C. Mining Methods at Nocozari, Sonora, Mexico, lxix, 1017.

LOCKE, AUGUSTUS. The Geology of the Tonopah Mining District, lxii, 217.

LUCAS, F. E. The Manufacture of Coke, lxxi, 1315.

LUCKE, CHARLES E. *Discussion* of Flameless Combustion, lxix, 965.

MACCALLUM, N. E. Recent Developments in Open-Hearth Steel-Practice, lxx, 1153.

MACDONALD, BERNARD. The Parral-Tank System of Slime-Agitation, lxiv, 387.

MAGUIRE, H. J. Replaceable Lips for Elevator-Buckets, lxix, 1017.

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Measurements and Relations of Hardness and Depth of Carbonization in Case-Hardened Steel. By MARK A. AMMON, lxx, 1167.

MEISSNER, C. A. *Discussion* of Flameless Combustion, lxix, 963.

Microstructure of Iron and Steel. By WILLIAM CAMPBELL, lxxii, 1357.

Mine Caves Under the City of Scranton. *Discussion* by RUFUS J. FOSTER, lxiv, 443.

Mine-Survey Notes. *Discussion* by E. R. Rice, lxiv, 447.

Mineral Production and Resources of China. By THOMAS T. READ, lxiii, 293.

Mining Industry of Japan. By KEIJERO NISHIO, lxi, 103.

Mining Methods at Nocozari, Sonora, Mexico. By D. C. LIVINGSTON, lxix, 1099.

Modification of the "Gay Lussac" Method for Silver-Bullion Containing Tin. By LOUIS EMLYNN SALAS, lxiii, 267.

MOORE, PHILIP N. *Discussion* of The Concentration of Iron-Ores, lxxii, 1562.

MURPHY, E. M. Electric Hoist at the Hecla Mine, lxix, 977.

MYRICK, C. M. An Experience in the Use of Water-Power, lxx, 1071.

NASON, FRANK L. The Bearing of the Theories of the Origin of the Magnetic Iron-Ores on Their Possible Extent, lxvii, 695.

New Electric Miners' Lamp. By D. B. RUSHMORE, lxvii, 739.

NICHOLAS, ASKIN M. *Discussion* of Slime-Filtration, lxviii, 887.

NISHIO, KEIJERO. The Mining-Industry of Japan, lxi, 103.

Note on the Case-Hardening of Special Steels. By ALBERT SAUVEUR and G. A. REINHARDT, lxxi, 1335.

Notes on the Laramie Tunnel. By DAVID W. BRUNTON, lxiv, 357.

Notes on the Laramie Tunnel. *Discussion* by W. L. SAUNDERS and H. FOSTER BAIN, lxvi, 659.

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Our National Resources and our Federal Government. By R. W. RAYMOND, lxx, 1111.

Parral-Tank System of Slime-Agitation. By BERNARD MACDONALD, lxiv, 387.

PAYNE, C. Q. Progress in Roll-Crushing, lxvi, 577; lxvii, 763.

Physical Data of Igneous Emanation. By BLAMEY STEVENS, lxiv, 429.

PORTER, JOHN JERMAIN. The Rational Valuation and Quality-Efficiency of Furnace-Stock, lxiii, 279.

PORTER, JOHN J. *Discussion* of The Effect of Alumina in Blast-Furnace Slags, lxxii, 1541.

Present Conditions in the California Oil-Fields. By MARK L. REQUA, lxiv, 377.

Present Conditions of Mining in the District of Vladivostok, Siberia. By A. F. J. BORDEAUX, lxx, 1025.

Present-Day Problems in California Gold-Dredging. By CHARLES JANIN, lxiii, 241.

Proceedings of the One Hundred and Second Meeting, New York, N. Y., February, 1912.

Proceedings of the One Hundred and Third Meeting, Cleveland, Ohio, October, 1912, lxxi, 1345.

Progress in Roll-Crushing. By C. Q. PAYNE, lxvi, 577.

Progress in Roll-Crushing. *Discussion* by E. G. SPILSBURY, C. Q. PAYNE, ROBERT H. RICHARDS, GEORGE C. STONE, S. ARTHUR KROM, lxxvii, 763; lxix, 592.

Rational Valuation and Quality-Efficiency of Furnace-Stock. By JOHN JERMAIN PORTER, lxiii, 279.

RAYMOND, R. W. Our National Resources and Our Federal Government, lxx, 1111.

RAYMOND, R. W. The Excursion to Hawaii and Japan, lxi, 1.

RAYMOND, R. W. *Discussion* of Alaska Coal-Land Problems, lxviii, 811.

RAYMOND, R. W. *Discussion* of The Refining of Blister-Copper, lxvi, 629.

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Refining of Blister-Copper. By HORACE H. EMRICH, lxvi, 611.

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REINHHARDT, G. A., and SAUVEUR, ALBERT. Note on the Case-Hardening of Special Steels, lxxi, 1335.

Replaceable Lips for Elevator-Buckets. By H. J. MAGUIRE, lxix, 1017.

REQUA, MARK L. Present Conditions in the California Oil-Fields, lxiv, 377.

RICE, E. R. *Discussion* of Mine-Survey Notes, lxiv, 447.

RICHARDS, JOSEPH W. The Schumacher Briquetting Process, lxv, 545.

RICHARDS, JOSEPH W. Treatment of Mine-Water from the Ashio Copper-Mine, lxvi, 655.

RICHARDS, ROBERT H. *Discussion* of Progress in Roll-Crushing, lxvi, 593.

RUSHMORE, D. B. A New Electric Miners' Lamp, lxvii, 739.

SALAS, LUIS EMLYNN. A Modification of the "Gay Lussac" Method for Silver-Bullion Containing Tin, lxiii, 267.

Sampling of Gold-Bullion. By F. P. DEWEY, lxx, 1075.

San Nicholás Mining District, San Nicholás, Tamaulipas, Mexico. By IRVING H. WENTWORTH, lxviii, 843.

SAUNDERS, W. L. *Discussion* of Notes on the Laramie Tunnel, lxvi, 659.

SAUVEUR, ALBERT, and REINHARDT, G. A. Note on the Case-Hardening of Special Steels, lxxi, 1335.

Schumacher Briquetting Process. By JOSEPH W. RICHARDS, lxv, 545.

SHIMER, PORTER W. Vanadium in Pig-iron, lxviii, 883.

Sintering and Briquetting of Flue-Dust. By FELIX A. VOGEL, lxv, 533.

Sintering of Fine Iron-Bearing Materials by the Dwight & Lloyd Process. By B. G. KLUGH, lxv, 507.

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Solubility in Nitric Acid of Gold Contained in Certain Copper-Alloys (Copper-Bullion). By EDWARD KELLER, lxvii, 681.

SPILSBURY, E. G. *Discussion* of The Concentration of Iron-Ores, lxxii, 1565.

SPILSBURY, E. G. *Discussion* of Progress in Roll-Crushing, lxvi, 592.

St. Helens Mining-District. By HORACE V. WINCHELL, lxx, 1037.

STEVENS, BLAMEY. The Laws of Igneous Emanation Pressure, lxiv, 411.

STEVENS, BLAMEY. Physical Data of Igneous Emanation, lxiv, 429.

STEWART, CHARLES A. The Geology and Ore-Deposits of the Silverbell Mining-District, Arizona, lxv, 455.

STONE, GEORGE C. Discussion of Progress in Roll-Crushing, lxvi, 594.

STOUGHTON, BRADLEY. Notes on Titanium and on the Cleansing Effect of Titanium on Cast-Iron, lxxi, 1245; lxxii, 1555.

STOUGHTON, BRADLEY *Discussion* of Measurements and Relations of Hardness and Depth of Carbonization in Case-Hardened Steel, lxxii, 1553.

STOUGHTON, BRADLEY. *Discussion* of Notes on Ruff's Carbon-Iron Equilibrium Diagram, lxxii, 1555.

Sulphatizing-Roasting of Copper-Ores and Concentrates. By UTLEY WEDGE, lxxii, 1523.

Surveying and Sampling Diamond-Drill Holes. By E. E. WHITE, lxxi, 1227.

SWEETSER, R. H. Blowing-In a Blast-Furnace, lxxi, 1327.

SWEETSER, R. H. *Discussion* of The Effect of Alumina in Blast-Furnace Slags, lxxii, 1541.

SWEETSER, R. H. *Discussion* of Notes on Ruff's Carbon-Iron Equilibrium Diagram, lxxii, 1555.

SWEETSER, R. H. *Discussion* on Recent Developments in the Inspection of Steel Rails, lxxii, 1571.

Ta-yeh Iron-Ore Deposits, Hu-pei Province, China. By C. M. WELD, lxx, 1059.

Titaniferous Iron-Ore Deposit in Boulder County, Colo. By E. P. JENNINGS, lxx, 1045.

Treatment of Complex Silver-Ore at the Lucky-Tiger Mine, El Tigre, Sonora, Mexico. By D. L. H. FORBES, lxviii, 769.

Treatment of Mine-Water from the Ashio Copper-Mine. By JOSEPH W. RICHARDS, lxvi, 655.

Vanadium in Pig-Iron. By PORTER W. SHIMER, lxviii, 883.

VAN HORN, FRANK R. The Occurrence of Silver, Copper, and Lead-Ores at the Veta Rica Mine, Sierra Mojada, Coahuila, Mexico, lxviii, 867.

VIVIAN, GODFREY T. Centrifugal Machines for Ore-Grading and Ore-Concentrating, lxx, 1103.

VOGEL, FELIX A. Sintering and Briquetting of Flue-Dust, lxv, 533.

WANJUKOW, W., and H. O. HOFMAN. The Decomposition of Metallic Sulphates at Elevated Temperatures in a Current of Dry Air, lxix, 889.

WEDGE, UTLEY. The Sulphatizing-Roasting of Copper-Ores and Concentrates, lxxii, 1523.

WELD, C. M. Ta-yeh Iron-Ore Deposits, Hu-pei Province, China, lxx, 1059.

WENTWORTH, HENRY A. Electrostatic Concentration or Separation of Ores, lxvi, 633.

WENTWORTH, IRVING H. The San Nicholás Mining District, San Nicholás, Tamaulipas, Mexico, lxviii, 843.

WHINERY, S. Clinton Iron-Ore Deposits in Kentucky and Tennessee, lxx, 1057.

WHITE, E. E. Surveying and Sampling Diamond-Drill Holes, lxxi, 1277.
WINCHELL, HORACE V. St. Helens Mining-District, lxx, 1037.
Wood Flotation Process. By HENRY E. WOOD, lxxi, 1227.
WOOD, HENRY E. The Wood Flotation Process, lxxi, 1227.
YALE, CHARLES G. Gold-Production in California, lxiii, 261.
YOUNG, GEORGE J. Fires in Metalliferous Mines, lxx, 1133.

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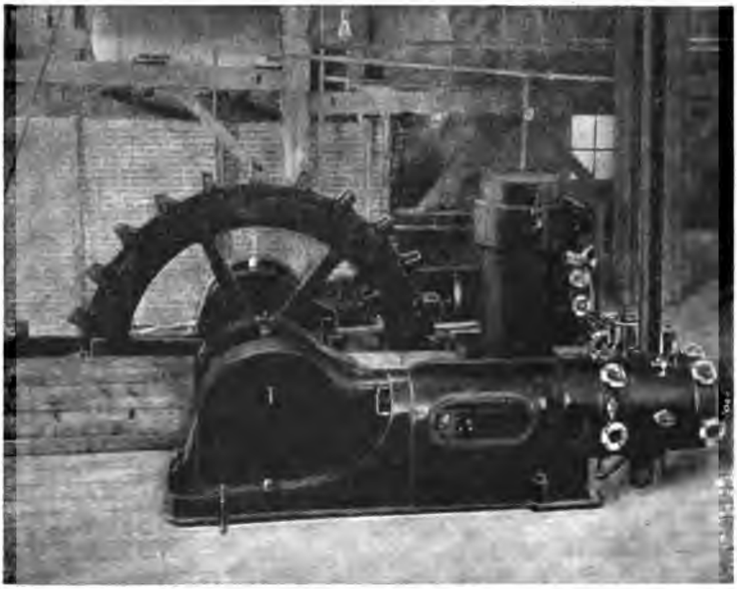

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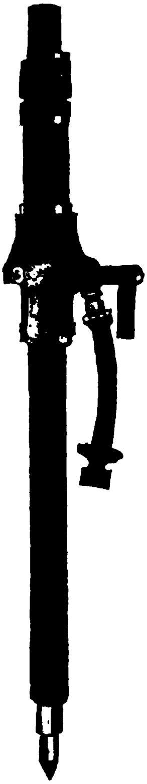

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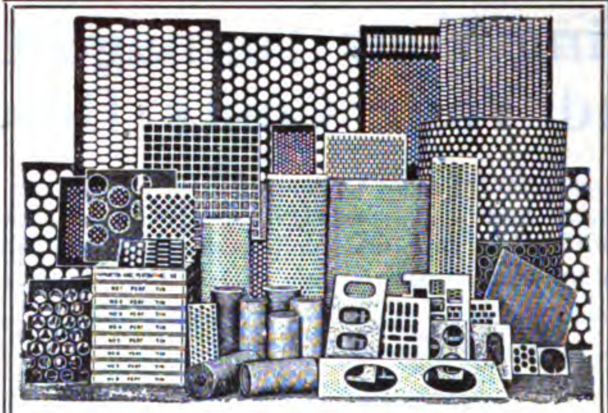

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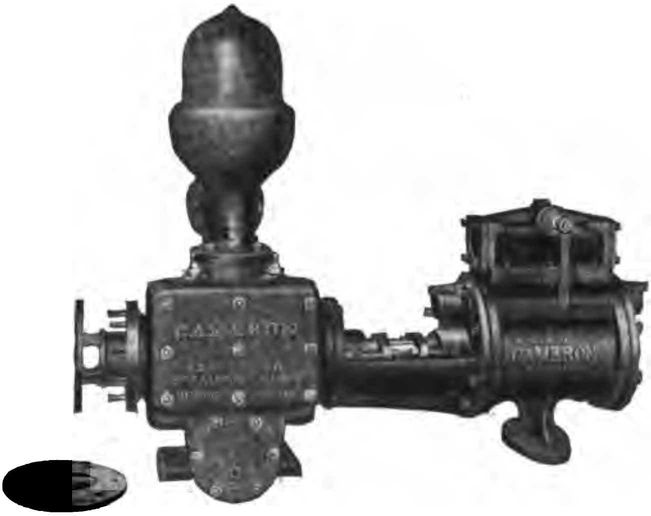

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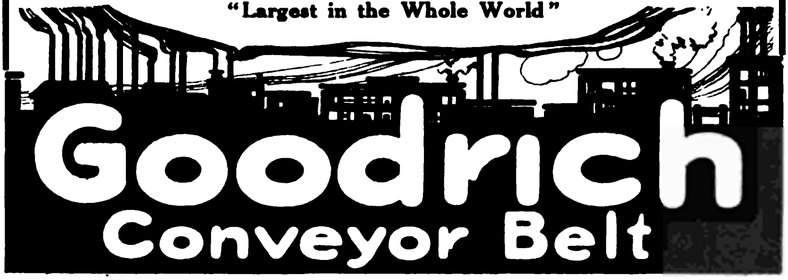

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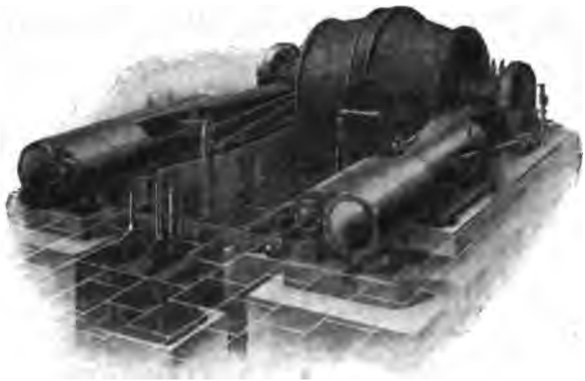

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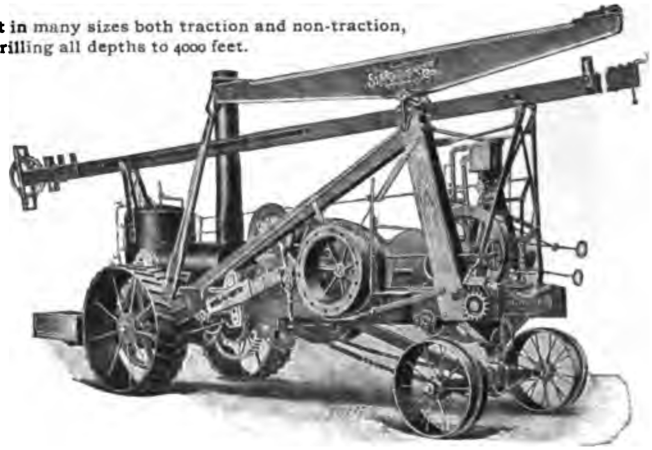

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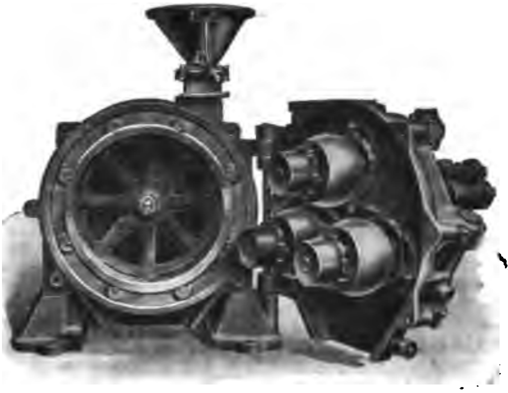

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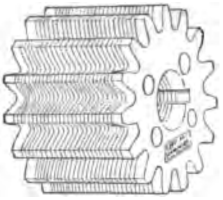

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A continuation of the "Posepny" Volume

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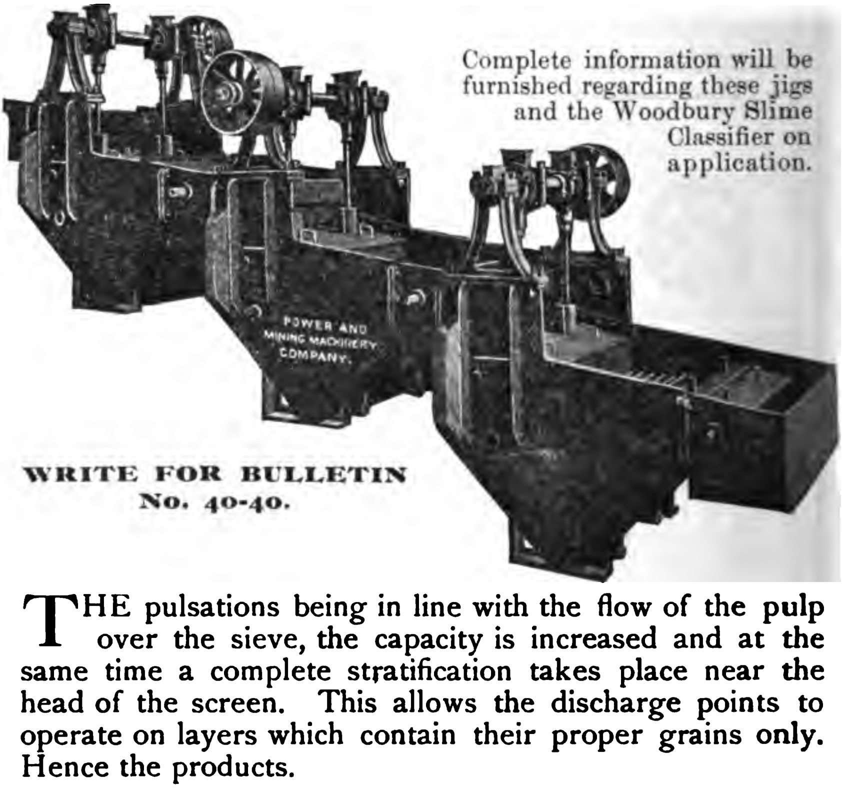

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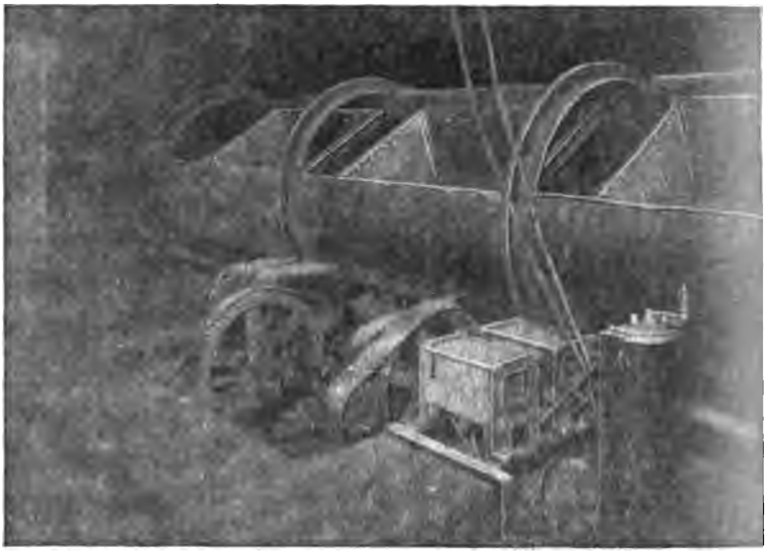

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